# Electrochemical and Metallurgical Industry

NEW YORK, DECEMBER, 1906.

## CONTENTS.

## Laws Governing the Promotion of Mining Companies.

In our November issue we showed how anxious the general public was to speculate in mining stocks, if once imbued with the idea that the cards were dealt fairly. Although there has been a great deal of endeavor on this part, nevertheless it has been fruitless of any widespread uniform legislation, but fruitful of an active public interest in the question. How far the work should be done by States and how far by the Federal Government is the great issue formulated by Jefferson and Hamilton a century and a quarter ago. At the same time the consideration is brought up how far a republic, which in its nature is individualistic, should lean towards paternalism.

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Undoubtedly some protection of the lambs against the wolves is necessary. But the endeavor should be in the line of educating the public so that the public protects itself. Any gross misstatement in the prospectus of a mining company is in essence an attempt to obtain money under false pretenses and thus a criminal offense—the offender to be prosecuted at the public expense. However, in the mining business it is so hard to estimate the "prospective value" of a mine, so hard to "look into the ground," that many a loophole would be left for the unscrupulous promoter. If we could always rightly discriminate between the true and the false, human life would become very tame. But something must be done to restrict the fleecing of the public by the use of advertisements and the mail, just as has been done by the national government in the case of the lotteries. A curb on those practicing the gentle art of swindling would be in order.

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A possible remedy suggests itself. Perhaps a national commissioner of mines with power to pass on mining enterprises offered the public, and to keep the statements to something like the truth, to eliminate pure "wildcats" and still leave the legitimate prospect, would help clear the atmosphere. But we frankly confess that this is a most difficult question, and the needed reform can only come as a result of much discussion. It should be remembered that the public are more willing to gamble on a good prospect if assured of a fair deal than to invest in a developed mine.

## Electric Steel, the Government and the Promoter

In another column of this issue will be found an abstract of an article from the New York *Sun* which deals with the unfortunate results which have followed the semi-official or unofficial newspaper reports of the government's experiments at Portland, Ore. We have already had occasion to refer to

these experiments and the erroneous impression they evidently made in the mind of the lay public. All misinformation is harmful, but the harm done is increased ten-fold when the erroneous information is apparently stamped with the government's approval. As is well known, one of the most wide-spread fallacies is the belief in the infallibility of governmental enterprises. In this case, the extravagance of the assertion that the government's experiments demonstrated that steel could be made directly from black sand at a cost of $12 per ton, is probably sufficient warning to the more intelligent layman that there must be a mistake somewhere. Therefore, the Black Sand and Gold Recovery Company shows much wisdom in addressing its appeal for subscriptions to the "plain people" and the "small investor," for in them is found the most profound belief and the most enduring faith in the infallibility of governmental work of this character.

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It is possible that if the results of the government's work had been first communicated to the public through the regular channels and had not been exploited in advance in the columns of yellow journalism, more conservatism would have been shown in their presentation. We have no doubt that when the official report of Dr. Day appears, it will be found to be quite a conservative document. The metallurgical purpose for which the Geological Survey's research was carried out—in which the electric smelting experiments formed only an incident—may then appear in a new light; and for this reason we hope that the appearance of the report will not be delayed too long. Even so, the question might still be fairly asked: What good end has been served by the government's experiments with the electric furnace? Any one with the most elementary knowledge of the electric furnace and the metallurgy of iron would know that the production of steel in a furnace like that used in the government's experiments is wholly impracticable. While it is true that a low-carbon iron may be produced in such a furnace, it is not practicable to manufacture in it those irons, having definite carbon contents, which we call steel. As for the production of pig iron in the government's furnace, it is extremely doubtful that any useful information could be thereby obtained. Experiments in this direction have been carried out on a comparatively large scale by others who possessed a far better equipment as regards apparatus and experience. Moreover, the results obtained are easily available.

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Among the opinions which have been expressed as to the object of the government's experiments, we notice one, stated with delightful frankness, in our Denver contemporary, *Ores and Metals*, which takes us to task for failing to perceive the true significance of the government's work. It is pointed out that Dr. Day's experiment "will make it less difficult to secure capital when conditions are ripe, and as a result many engineers who have known all about the facts for years will be given employment." It would be difficult to gainsay this in view of the *Sun's* article; but in spite of the astonishingly rapid growth of the government's ubiquitous activities, we doubt if any department is as yet designed to assume the duties of a promoter of stock companies or an engineers' employment agency.

## The Metal Markets.

As was clearly indicated in our July editorial on this subject, the fall metal markets have followed the precedent of the last year and displayed unusual strength after early buying. The demand for metals results from basic reasons for expansion. Solid and substantial buildings and factories require metal in all their parts. New factories require new machines. Everywhere is iron, steel, copper, zinc and lead required in new construction. The electrical business in all its complexities requires chiefly iron and steel, but copper most of all. It also uses zinc for primary batteries and lead for storage batteries and for lead-covered underground cables. But the rapid expansion of all the electrical trades has put the copper producers under tremendous physical strain for simply quantity. The pig iron and steel markets are wisely held at constant points by the denominating interests, but of late have shown a great natural strength. Possibly the statistical position in the iron and steel markets is the strongest at the present time.

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Lead market is especially strong, for the scarcity of lead ores is most actual. Spelter is in great consumptive demand, and has risen forty points since September. The presence of several smelters with inadequate financial resources tends to hold spelter down. However, the great demand for galvanized sheets, due to high prices of lumber, for which they are a substitute, and the tendency to put up substantial structures, as well as the demand on the spelter producers by the other galvanizing industries and the brass trade, have given the spelter market great basic strength. Considered in their entirety, the metal markets are all in good shape. The demand, though at times excited, is in main healthy, because of the great growth of this country. The mining and metallurgical engineers are certainly doing their share for the good of the country in producing the metals which are the framework of industrialism.

## Progress in Reverberatory Copper Smelting

In the past three years there has been a marked tendency to revert to the reverberatory copper furnace for the purposes of smelting copper ores or concentrates with the production of a copper matte carrying the precious metals. While the vertical blast furnace has made great progress, especially in the "pyritic" line, nevertheless, the open flame reverberatory has made greater irrelative advances. The rivalry is a survival of the old struggle between the "Welsh" and "German" style of smelting described in both the works of "Percy" and "Peters."

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The shaft furnace has the advantage of continuity of operation and the reduced labor charge resulting therefrom. While its fuel is more expensive, yet the proportion of fuel to charge is so much less that with the combustion of the sulphur and iron of the charge the advantage in this respect is on the side of the Teutonic method. The cost of repairs is also less. Here the superiority of the shaft furnace ceases and that of the reverberatory commences. Perhaps the latter's greatest advantage lies in the fact that its flue dust losses are nearly nothing, while in the other style the losses are considerable with high blast pressures now used, and with large proportion of the flue dust necessarily recharged. This results in a cycle of

"fines," and thus increases smelting charge. In the matter of slags the Welsh method is far superior. With a proper mechanical mixture there is no danger of freezing, for the heat of the furnace can be regulated at will—an innate point of superiority of the intermittent type. There is also a far wider latitude in slag composition, which circumstance was used to advantage in "Highland Boy" plant in the Bingham district. Due to the fact of high heat and large pool, more or less quiescent, the slags are usually cleaner. There are possible places where the reverberatory smelting practice can be improved, such as by the use of the "Eldred" process for delaying combustion and the return of part of the sensible heat of chimney gases now used for raising steam by recuperators and by improved refractory brick for arch.

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The reverberatory smelting process was mainly introduced into this country by a man with Welsh experience, Prof. Richard Pearce, who used it with great success at the "Argo" plant in Denver. His furnaces once considered large are now small compared to the mammoth reverberatories of the Anaconda "Washoe" works. The installation of reverberatories is rapid, and the "Phelps-Dodge" interests and the Nevada Consolidated Copper Co., among others, are taking active interest in this subject. It will be used largely for treatment of roasted fine concentrates and "flue dust" from shaft furnaces or in localities where coal is cheap and coke dear, on ores which when mixed produce a slag of high "fluidity-point." Put in its proper place or combined with the shaft furnace it is a highly efficient metallurgical instrument for extracting values from copper, silver and gold ores.

## A National Department of Mining and Metallurgy

One of the marked facts bearing on our commercial life to-day is the increased yield per acre of corn, wheat, cotton and other staples in the great agricultural States. This in part is due to the better financial position of the farmers, but it is more due to the diffusion of agricultural science. The Department of Agriculture at Washington spends some seven million dollars each year in scientific and practical research. The different State agricultural stations, connected often in the West with a magnificent State university, supplement the work. In farming, the base of all wealth, for we are all dependent on our stomachs, the ideals of the twentieth century, the application of broad truths of science to the amelioration of human life are manifest. And the direct realization of these ideals is accomplished through a great national system for the acquiring of new knowledge and the distribution of information. Although this may not have the effect of doubling the yield per acre in less than ten years, the enthusiastic claim of Secretary Wilson, yet it is a fact that it is making now a national saving that is measured in the hundreds of millions of dollars.

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Next to farming, the mining of ores and getting their valuable contents in marketable shape is the most important basic source of wealth. While there is much done by the United States Geological Survey and the Commissioner of Mines or State geologist in the several States, yet the organization is far from perfect and not to be compared to Secretary Wilson's

splendid force. The work of the "Coal Testing Plant" of the United States Geological Survey, at St. Louis, is an example of what might be done in all forms. The tests on gas producers working on lignites have shown possibilities in the Western coal deposits of the tertiary ages.

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While the work at St. Louis is good, yet it has been necessarily of a restricted nature and in only one industry. The smelting of iron ores, the manufacture of steel, the smelting of copper ores and the refining of copper, are great branches of the mineral industry. In these and in the treatment of zinc, lead, silver and gold ores the elimination of waste and the recovery of by-products are being accomplished, but by a slow and painful progress. If all the attempts at progress were reported and filed at Washington, if systematic scientific investigations were made of the chemical reactions of the different processes, if all this were done by one great bureau with a view to the practical application, the increased efficiency in mining and metallurgical operations would be astounding, because it is easy to make changes when comparatively few men's minds must be changed. Of course, the consolidation of experience and the direction of research when done at Washington is a work of great magnitude. It would be supplemented by the work at the universities and technical schools and engineering societies, as well as by the research of the engineering staffs of the large corporations, and if rightly laid out would not interfere with the endeavor of any one. But it would rather prevent the reduplication of experiments and would co-ordinate metallurgical experience.

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Canada has done just about what we have outlined, but it is easier to effect such a result in the smaller country. The admirable reports on electric iron smelting and steel making, and the later report on the zinc resources of British Columbia, which we reviewed in our November issue, are examples of the good work done by the Dominion Government in this line. Germany also is most provident in her utilization of national resources, but her governmental direction and interference would not be allowed in this country. Anyhow, her celebrated Physikalisch-Technische Reichsanstalt has found successors in this country and in England, and now a Chemische Reichsanstalt is earnestly contemplated. The chemical industries represent perhaps the most flourishing commercial activity of Germany, and this great national success has been essentially due to the fact that for the German chemical manufacturers commercial chemistry has been an applied science. Their most valuable raw material was and is the army of trained chemists turned out every year by the German universities. Under existing conditions, German chemists naturally favor the manufacture of the more complicated products of organic chemistry. Reversely, in this country, with its abundance of valuable ore resources, applied chemistry will naturally remain metallurgy for years to come, though in future metallurgical developments will probably be guided more systematically by scientific principles than has been the case in the past. At all events there is a crying need in this country for a great national bureau to take over and amplify the work of the Geological Survey, and increase the operating efficiency of the processes of extracting economic values out of the ground.

## The Iron and Steel Market.

Buying of pig iron and finished steel products was relatively light during November. The market had been well sold up and the very heavy buying could not be continued indefinitely. The chief interest now is in specifications against old contracts and the success the furnaces and mills meet in endeavoring to fill them.

As was expected, October established new records of production in a great many departments and aided by good weather the pace was maintained in November. Production of coke and anthracite pig iron in October was 2,196,808 gross tons, a gain of 23,000 tons over March, 1906, the previous record month. The furnaces controlled by steel companies made a greater gain than this, the merchant furnaces falling some 30,000 tons behind. The furnaces operating in March did not make as much pig iron in October, the gain being due to the completion of new furnaces. As more of the older furnaces are brought into the producing class, new high records can be made. We estimate the production of all grades of pig iron in 1906 at about 25,300,000 tons, but the furnaces now completed can make a considerably larger tonnage in 1907, and there are other furnaces now nearing completion.

There being no considerable stocks of steel-making pig iron, production of steel has been equally large. The Illinois Steel Co., for instance, made in October, 212,547 gross tons of steel ingots, against its best previous record, made in September, of 192,385 tons. Individual finishing mills have not shown a proportionately large number of new records, for the reason that all finishing mills have been full of business, and the steel supply had to be distributed. When only a portion of the finishing mills demand the greatest output they can be better supplied with steel. Finishing capacity is always in excess of steel capacity.

### The Ore Situation.

On Friday, Nov. 2, an informal arrangement was made by ore sellers as to prices on Lake Superior ores for the forthcoming season. Nominal advances were made of 75 cents on Bessemer ores and 50 cents on non-Bessemers, but the base iron content, maintained without change for years, was reduced, so that the actual advances were larger than the nominal. The reduction in the base was forced by the increasing leanness of the ores. In the old days only the choicest deposits were mined. As demand increased, less desirable deposits had to be mined, while the older deposits are becoming exhausted. It may be estimated roughly that the average iron content of all Lake Superior ores shipped is decreasing by between 1 and 2 per cent per year at the present time. No other outcome could be expected. Shipments this year will be in the neighborhood of 37,000,000 tons, and predictions for next year are at 40,000,000 tons and more. In 1899 they amounted to only 18,251,804 tons, and in 1890 and 1892 they were a trifle over 9,000,000 tons, the years here mentioned having been record years in their time, so that the output has doubled twice in sixteen years.

Until last year the basis was uniform as follows: On Bessemer ores, 63 per cent iron when dried at 212°, and 10 per cent moisture in the natural state, making the iron content, natural, 56.7 per cent; on non-Bessemers, 60 per cent iron, when dried at 212°, and 12 per cent moisture in the natural state, making 52.8 per cent iron in the natural state. Last year one merchant firm departed from the custom and guaranteed 53 per cent iron on Mesabi non-Bessemers, the innovation being accepted by most sellers. In establishing a new basis even figures were selected for the natural, instead of the dried state, although the moisture content is unchanged, the new bases being, in the natural state, Bessemers, 55 per cent; non-Bessemers, 51.5 per cent. In the following table is shown the 1906 season price, the 1907 season price, on the new basis, and the price which an ore of the previous guaranty would bring on the 1907 price basis:

| | 1906 Base. | 1907 Base. | 1906 Base at 1907 Price. |
|---|---|---|---|
| Old Range Bessemer | $4.25 | $5.00 | $5.154 |
| Mesabi Bessemer | 4.00 | 4.75 | 4.897 |
| Old Range non-Bessemer | 3.70 | 4.20 | 4.306 |
| Mesabi non-Bessemer | 3.50 | 4.00 | 4.116 |

These prices are f. o. b. lower lake port. Contracts are usually made for a season's supply, to be shipped during the season of lake navigation, while payments are made in twelve equal monthly installments, May 1 to April 1. Only a comparatively small percentage of the Lake Superior output is sold on this basis; more than half the ore is mined by consumers, while a considerable part of the balance has been sold on long-time contracts.

Most consumers promptly bought ore on the announcement of prices. A few have held off, being unwilling to commit themselves so far ahead, as the consuming season for ores shipped next season runs to about July 1, 1908.

### Pig Iron.

The market has continued to move upward since last report. The pressure for prompt delivery continues as the great increase in production, although aided by imports at the rate of 10,000 to 20,000 tons per week, has not filled the demand, and prompt lots are bought at higher and higher prices. Furnaces being well sold through first quarter, and in many cases also through second quarter, are disposed to exact prices for such deliveries pretty closely in line with prices for prompt. For later delivery the market is naturally on a lower basis. Buying is not heavy. For prompt shipment we quote foundry iron at close to $25, valley furnace, and $22 to $23, Birmingham. A few sales of Bessemer and basic have been made for prompt and first half delivery at from $21.50 to $22.60, valley, while for second half the market on these grades is about $21, valley, although no large sales have been made.

### Billets and Sheet Bars.

There is no regular market on billets. All consumers are covered more or less fully by contracts and only buy as they can supplement their deliveries by odd lots at reasonable prices. Sales have been made at somewhat higher prices, and Bessemer billets may be quoted at $29 and open-hearth at $30 to $32. Sheet bars are about $30, Pittsburg, for long lengths.

### Finished Material.

Since last report the following advances have been made:

Tin plates, 15 cents a box to $3.90.

Black sheets, $2 a ton to $2.60 for 28 gauge.

Galvanized sheets, $2 a ton to $3.65 for 28 gauge.

Blue annealed sheets, $1 a ton to $1.80 for 10 gauge.

Painted corrugated roofing 10 cents a square to $1.85.

Galvanized corrugated roofing 5 cents a square to $3.15.

Wire products $1 a ton to $1.90 for wire nails, $1.75 for plain wire and $2.35 for galvanized barb wire.

Light rails $1 a ton to $32 for 20 to 45-pound sections.

Merchant steel bars $2 a ton to $1.60

An advance in plates from the $1.60 basis is likely to be made at any time to $1.70. The Eastern mills about the middle of November began quoting on a basis of $1.70, Pittsburg.

Beams and channels remain at $1.70 for 15-inch and under and $1.80 for larger sections, and are not likely to be advanced.

## Electric Steel

The New York *Sun* of Nov. 21 publishes an article on steel direct from ore, covering almost four columns of its editorial pages.

The article is different from anything that has been pub-

lished in daily newspapers on electric steel. While sensational in its arraignment of the scheme of the Black Sand & Gold Recovery Co., the article appears to be extremely careful in the statement of facts, and is at the same time conservative in the best sense of the word in the discussion of the possibilities of electric furnace methods in iron and steel metallurgy. The anonymous writer in the *Sun*, who is evidently an electric furnace expert, intimately connected with the iron and steel industry, expresses the following "safe and sane" views on the possibilities of making iron and steel in the electric furnaces.

Concerning competition of the electric furnace with the blast furnace for pig iron production, he says that such competition is impossible where coke is cheap. Only under exceptional conditions where electric power is cheap and coke dear it may be possible to produce pig iron at a lower price than the blast furnace product can be marketed.

Concerning the manufacture of steel in the electric furnace, reference is made to two types of furnace which have proven commercially successful. One is that of Héroult and Keller; the other the induction furnace of Colby and Kjellin. For the manufacture of high-grade tool steels (such as are now made by the crucible process) the electric furnace has already distinct advantages, and it is possible that, with the development of the art, the electric furnace may in time also encroach upon the field of the open-hearth furnace.

The main portion of the article in the *Sun* is, however, devoted to a severe criticism of the methods employed in the exploitation of a scheme dealing with the production of steel direct from ore. The article quotes largely a prospectus issued by the Black Sand & Gold Recovery Co., with head offices in Chicago, which prospectus bears on its front cover the following statement:

"This pamphlet tells how the Government of the United States has recently made steel in commercial quantities for $15 per ton less than the Steel Trust can make pig iron." The Black Sand & Gold Recovery Co. proposes to work on the black sand on the Pacific Coast, this sand containing considerable quantities of magnetic oxide of iron, and to produce steel in electric furnaces. The principal basis of its appeal for subscription to its stock is certain work carried on recently under the auspices of the United States Government.

In connection with the black sand deposits of the Pacific Coast, Congress made an appropriation of $25,000, afterward increased by a second appropriation to $50,000, for their investigation. This investigation was carried out under the supervision of Dr. David T. Day, chief of the Division of Mining and Mineral Resources, United States Geological Survey. The complete official report by the government on this work has not been issued, but certain semi-official articles have appeared giving some information as to the government work. These articles have chiefly appeared in daily newspapers and their general tone is highly sensational.

Special reference is made to an article in the issue of the Chicago *Inter-Ocean* of Sunday, Jan. 21, 1906. This article has received the endorsement of Dr. Day in a letter to the business manager of the *Inter-Ocean*, which is reprinted in the Black Sand & Gold Recovery Co's prospectus. In this letter Dr. Day says that the article in the *Inter-Ocean* "requires but little in order to make it perfect. * * * The black sand subject is really one of very great interest, and is going to aid very much good citizenship on the Pacific Coast, for it is not a matter of speculation, but simply of untiring industry, with all the personal improvements of character which come by that kind of work as contrasted with the usual speculation so frequent in the mining industry."

From this article in the *Inter-Ocean* the writer in the *Sun* quotes the following passages:

"Dr. Day, after demonstrating that many of the sands tested contained as high as 600 pounds of iron to the ton, erected a 10-ton electric furnace, and in one short hour, by adding lime and broken coal to the iron which had been separated from the black sands, smelted the iron into high grade steel." The immense importance of the steel industry on the Pacific Coast is discussed, the construction of "battleships from sand" is prophesied, and so forth. Finally, the article is summarized in a series of results which "Dr. Day and the work of his associates have demonstrated beyond question." Among these results are the following:

"That in order to utilize this iron (magnetite) commercially it must first be magnetically separated from the sand without drying."

"That the magnetic iron (oxide?) when separated from the sand can be reduced in an electric smelter to commercial iron and steel ready to supply Pacific Coast markets."

"That the steel so produced, smelted with cheap electricity generated from water power, need not exceed $12 per ton, while the present cost of pig iron on the Pacific Coast is over $27 per ton—a saving of $15 per ton over present prices of pig iron alone."

These claims are most severely criticised by the writer in the *Sun*, who refers to an article in the *Technology Quarterly*, written by Prof. R. H. Richards, who was associated with Dr. Day in the black sand investigation. There Prof. Richards states that the furnace "is still in the experimental stage, as to the means of controlling the contents of the steel in silicon carbon, manganese, sulphur and phosphorus." The writer in the *Sun* points out that it is obviously absurd to talk of manufacturing steel when the contents in carbon, silicon, etc., cannot be controlled.

Many investigators have made extended experiments to make steel directly from the ore in one operation, but all have failed so far in solving this problem, and "we may feel confident that this was not done in the crude experiments made by the Government at Portland," especially if the type of furnace used in these experiments is considered, which is in no way suitable for making steel. "Under these circumstances it would be a waste of time to discuss the possibility of turning out steel from an electric smelter at $12 a ton."

Concerning magnetic separation, the writer in the *Sun* makes the following criticism:

"Now, as regards the magnetic separation of the magnetic oxide of iron from the black sand the Black Sand & Gold Recovery Co. publishes a letter, apparently written on the United States Geological Survey's paper by Mr. J. F. Batchelder, who is now the company's superintendent, in which Mr. Batchelder asserts that as a result of certain tests made at the request of Prof. Richards, it was found that the Lovett magnetic concentrator was a most excellent machine. This machine is said to be controlled by the Black Sand Co. It is interesting to note that in a letter, signed by Dr. Day and printed in the *Engineering and Mining Journal*, the work with Prof. Richards is referred to, and it is asserted that no machine owned by the Black Sand & Gold Recovery Co. was tested. In another part of his letter Mr. Batchelder says that one of the strong features of the Lovett machine is its ability to handle the material wet. Thus it is not necessary to dry the sand, 'which must be done in the case of every other separator.' This is absolutely false, for there are other wet separators."

In taking up again the pamphlet of the Black Sand Gold Recovery Co. the writer in the *Sun* says: "But it is not only the fatuous experiments of the Government's employees that are used to excite the plain people to buy stock in the company, but various forms of subtle misrepresentation are also employed. For example, the prospective investor is inclined to infer, after reading the literature furnished him by the company, that one C. E. Wilson, who is described as 'the foremost furnace electrical expert of this country' is T. L. Wilson, whose name is intimately associated with the development of the calcium carbide industry in America. Even if the blame for this misapprehension can be shifted from the company to its

dupes, no excuse can be found for another ambiguous suggestion. One of the illustrations in the company's pamphlet shows 'pig iron melted in the electric furnace,' and the reader is led to infer that this pig iron was manufactured by Dr. Day and his associates or by the Black Sand & Gold Recovery Co. The illustration actually had its origin in a photograph of pig iron taken during the experiments made under the auspices of the Canadian Government in Sault Ste. Marie during January and February and March of this year. Perhaps reference to those experiments is avoided because from the results it has been estimated that under favorable conditions crude pig iron might be produced at about the cost advertised by the Black Sand & Gold Recovery Co. for finished steel. But then such facts as this would be of no assistance in the sale of stock to gullible 'plain people.'"

"An air of stability is given to this questionable enterprise by the fact that on the Pacific Coast are found some of the conditions that favor the use of the electric furnace, such as a high price for fuel, possibilities for cheap development of water power and somewhat higher prices for finished products than are found in the East. Probably it is considerations such as these which have induced such well conceived enterprises as that of Mr. Noble, of the Northern California Power Co., who is putting up an experimental electric furnace plant, to make pig iron from the ore of the Shasta Iron Co." (Concerning this undertaking, in which the Héroult furnace will be used, probably in essentially the same way as in the Sault Ste. Marie experiments, some information was given in our November issue, page 441.)

The writer in the *Sun* concludes as follows:

"It is lamentable that just when the electric furnace is emerging from the laboratory into a useful metallurgical apparatus it should have to bear the odium of such irresponsible wildcat exploitations as that of the Black Sand & Gold Recovery Co. But it is still more deplorable that such schemes should find a basis in the inexpert activity of governmental bureaus."

We comment on the article on another page.

# CORRESPONDENCE.

## Secretaryship of American Electrochemical Society.

*To the Editor of Electrochemical and Metallurgical Industry:*

Sir:—As Secretary of the American Electrochemical Society, I desire, through your columns, to lay before the membership of the Society a matter that has given me some thought for months past. Although the duties of the office of Secretary have been very congenial to me, and the acquaintances and friendships made while occupying the post, very greatly appreciated, I feel that I will have to decline to be a candidate for re-election at the next annual meeting. To do the work justice requires constant thought as well as work, and the growing demands of my professional business are now pressing for more of my time than is available under the present arrangement.

I thought it best, therefore, to give this timely notice, that the members might find some one for the position who could keep in touch more constantly and thoroughly with the lines of research and industrial development in the field of electrochemistry. In this way the high standard of the Transactions and membership may be kept up and the well being of the Society in general promoted.    Samuel S. Sadtler.

Philadelphia, Pa.

[Prof. J. W. Richards, of Lehigh University, declared in a letter to this journal, February, 1904, when a similar question arose as to the secretaryship of the Society, that "he is willing to serve the Society in any capacity in which the members think he can best aid it and promote its interests." We feel sure that Dr. Richards has not changed his attitude towards the Society,

and that his acceptance of the secretaryship would probably represent the best possible solution of the problem now confronting the Society.—Editor.]

## The Mining Boom.

*To the Editor of Electrochemical and Metallurgical Industry:.*

Sir:—I have read with interest the editorial on the "Mining Boom," in your November issue, and especially that part which refers to the increase in the values of mines due to the advance in the art of metallurgy.

I could have wished that you had made particular reference to the zinc business. To my knowledge, two zinc properties, the "Kelly" mine, in New Mexico, and the "Iron-Silver" mine, in Leadville, went begging, the latter four years ago, the other two years ago. The figure the mines were offered for was in each case less than $100,000. Later, advance in handling complex sulphide ores by modern methods of concentrating and in handling zinc ores high in iron by Kansas zinc smelters has brought it about that these mines are each netting several hundred thousand dollars per year, and are worth a good many million dollars besides. I mention these two, for they are the most conspicuous examples in the zinc business; of course, there are many similar instances. The list might be extended greatly, and in the future we can expect similar great enhancement in the value of mines now considered worthless, due to "the advance in the state of art."

New York City.    Woolsey McA. Johnson.

## Specific Heat of Iron.

*To the Editor of Electrochemical and Metallurgical Industry:*

Sir:—I do not know whether your readers have yet seen the very valuable determinations of the specific heat of iron by Mr. Harker[1]. They are of particular interest in that they confirm in general the early results obtained by Pionchon, when the latter are corrected in accordance with our present values of some of the constants which he used.

I enclose a copy of Mr. Harker's results, together with those of Pionchin as originally given and as recalculated by Mr. Harker:

TOTAL HEAT OF IRON BETWEEN $O^0$ AND $T^0$.—$O_0^T$.

| T. | Pionchon's Values from Formula. | Pionchon's Values Recalculated. | Harker's Results. |
|---|---|---|---|
| 200 | 23.5 | 23.5 | 23.5 |
| 300 | 36.8 | 36.8 | 37.0 |
| 400 | 51.6 | 51.6 | 51.3 |
| 500 | 68.2 | 66.0 | 66.9 |
| 600 | 87.0 | 83.2 | 83.8 |
| 700 | 108.4 | 102.2 | 104.1 |
| 800 | 135.4 | 125.0 | 127.8 |
| 900 | 157.2 | 146.7 | 148.0 |
| 1000 | 179.0 | 166.0 | 155.7 |
| 1100 | ..... | ..... | 168.8 |

MEAN SPECIFIC HEAT BETWEEN $O^0$ AND $T^0$.—$S_0^T$.

| T. | $S_0^T$. | T. | $S_0^T$. |
|---|---|---|---|
| 200 | .1175 | 700 | .1487 |
| 250 | .1204 | 750 | .1537 |
| 300 | .1233 | 800 | .1597 |
| 350 | .1257 | 850 | .1647 |
| 400 | .1282 | 900 | .1644 |
| 450 | .1311 | 950 | .1612 |
| 500 | .1338 | 1000 | .1557 |
| 550 | .1361 | 1050 | .1512 |
| 600 | .1396 | 1100 | .1534 |
| 650 | .1440 | .... | .... |

Dr. Harker's paper appeared in the *Philosophical Magazine* for October, 1905.    Henry M. Howe.

Columbia University.

[1] An abstract of Dr. Harker's paper appeared on page 436 of our vol. III. We are glad, however, that Prof. Howe has been good enough to again call attention to this work, and to send us the above tables, which are fuller than those published before in our columns on this subject.—Editor.

## The Extraction of Metallic Sodium.

### By C. F. CARRIER, JR.

#### (Concluded from page 446.)

ELECTROLYSIS OF FUSED SODIUM CARBONATE, 669.733.374.3.

Only the process of H. Becker can be classed in this group. This inventor claims to increase the efficiency, decrease the cost and obviate explosions by using a mixture of sodium carbonate and sodium hydroxide as the electrolyte. He assumed that the carbonate would be decomposed in the presence of the hydroxide. This process is carried out in an apparatus that is very similar to the Castner apparatus, except that there is no metal screen diaphragm between the anode and the cathode, and the metal is collected in a water-cooled collecting hood. A full description of the process and apparatus has already been published in this journal, and it has also been shown by the author in an investigation undertaken in the laboratory of Prof. Max Le Blanc, that the fundamental assumption of the process is false, no carbon dioxide being liberated by the electrolysis of a mixture of sodium carbonate and hydroxide. If a pure bath of carbonate is used the melting point is so high that the sodium liberated will reduce the carbonate, leaving free carbon in the bath. (Private communication from Prof. Haber's laboratory, Karlsruhe.)

References:

This Journal 1; 574 (1903) and 2; 357 (1904).

U. S. P. No. 663,719 of Dec. 11, 1900.

Also patents in Eng., Ger., Fr. and Switz.

ELECTROLYSIS OF FUSED SODIUM HYDROXIDE, 669.733.734.4.

In this group come the only processes that, up to the present time, have been commercially successful in producing sodium. Like natural phenomena, invention first tends to follow the line of least resistance, so it is quite natural to expect the first successful solution to be by use of the easily fusible sodium hydroxide. Although the reduction of sodium from sodium hydroxide has been in commercial use for a number of years, the exact mechanism of the chemical reactions is still a matter of controversy.

It was observed by Janeczek in 1875 that both oxygen and hydrogen are obtained by electrolysis of an anhydrous bath of fused potassium hydroxide. He also noted that water condensed in his gas receivers, and concluded that OH ions must have been discharged at the anode. The evolution of hydrogen he attributed to the action of potassium on potassium hydroxide. It seems more reasonable to assume, however, that the reaction must be between water and potassium, for the OH ions decompose as soon as discharged, forming water and oxygen. A portion of the water escapes as vapor, as noted above, but all the water cannot be expelled by heating either potassium or sodium hydroxides to their melting points, or slightly above. This portion of water remaining in the bath will, therefore, be decomposed by the current in preference to the alkali base, water requiring but 1.69 volts, while sodium hydroxide requires 2.2 volts. It would, therefore, seem probable that a portion of the hydrogen at least is produced primarily by electrolysis.

Reference:

Berlin, Ber. 8; 1,018 (1875).

The electrolysis of fused sodium hydroxide with a cathode just touching the surface of the bath was investigated by Lorenz and Sacher in 1901. They made determinations of the voltage of decomposition a problem, which was more thoroughly investigated by Le Blanc and Brode, although the results of both substantially agree. It was found that the voltage of decomposition for anhydrous sodium hydroxide is 2.2 volts, but when water is present there is also a lower decomposition point at 1.3 volts. The formation of water from two OH ions at the anode was also demonstrated, and at high temperatures it was noted that hydrogen also was produced at the

anode. This they attributed to the action of sodium that had diffused from the cathode to the anode, where it reacted with the water formed from the OH ions.

References:

Lorenz and Sacher, Zeit. f. anorg. Chem. 28; 385 (1901).

Lorenz, Zeit. f. El-ch. 8; 873 (1902) and 9; 155 and 333 (1903).

Le Blanc and Brode, Zeit. f. El-ch. 8; 697, 717, 817 and 939 (1902), and 9; 230 (1903).

Hambuechen has shown that with aluminium electrodes it is possible to separate sodium from sodium hydroxide by means of an alternating current; Haber has shown the electrolysis of the same compound in the solid state.

References:

Hambuechen, Trans. Am. El-ch. Soc. 4; 105 (1903).

Haber, Zeit. f. anorg. Chem. 41; 407 (1904).

As early as 1866 an English patent (Dickson, No. 2,266 of 1866) specified the use of sodium hydroxide as a raw material in the production of sodium, but it remained for H. Y. Castner to devise the first practical electrochemical process for the production of sodium. His process has been in continuous
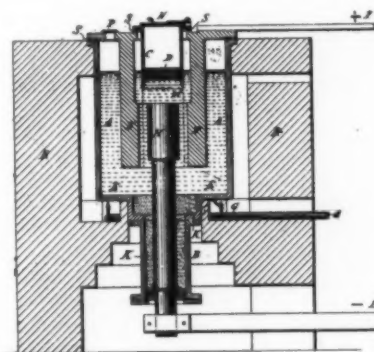


FIG. 5.—CASTNER CELL.

commercial operation from the time of its first appearance until now. The process consists in electrolyzing fused caustic soda in a specially constructed cell, at not more than 20° C. above its melting point. This cell consists (Fig. 5) of an iron melting pot A, suitably mounted in masonry. The cathode H is of metal, and is introduced into the cell through the bottom by means of the sleeve-like extension B, which is filled with solid electrolyte to insulate the cathode from the pot, hold the cathode in place and seal the joint. The anode F is an annular mass of metal surrounding the cathode, and it is supported from the cover. Suspended above the cathode is a cylindrical iron vessel C, whose lower edge supports a cylinder of iron or nickel gauze that completely surrounds the cathode and separates it from the anode. The sodium is collected in the vessel C. For full description of the Niagara Falls plant using this process, see this journal 1; 15 (1902).

The advantages of this process are:

(a) Low melting point of the bath.

(b) Simplicity of the apparatus.

The disadvantages are:

(a) High cost of raw material.

(b) Low ampere efficiency.

(c) Small size of the units.

(d) Constant explosions during process.

The two advantageous factors tend to decrease the wear and tear and thus the maintenance charges. Sodium hydroxide (78 per cent $Na_2O$) costs about $45.00 per ton, while an equivalent quantity of salt is worth about $6.00. No by-products are obtained in the electrolysis of sodium hydroxide, while the chlorine from the electrolysis of salt can be worked up with greater or less profit according to current conditions. The

ampere efficiency is theoretically only 50 per cent, since for every atom of sodium liberated an atom of hydrogen must also be formed. In practice the efficiency is probably not over 40 per cent, and by some authorities is placed as low as 35 per cent. Another great drawback is the small size of the units. Owing to the constant explosions of oxygen and hydrogen in the cell, it would be dangerous to make the cells of any great size. This tends to lower the efficiency and increase the charges for maintenance and attendance. The metal is removed in dribblets by dipping out with a spoon, so the latter item may be a considerable factor.

For comparison with a process to be discussed later, the following estimate is made of the cost of producing sodium by the Castner process under the following conditions: Daily production, 1 ton of metallic sodium; ampere efficiency, 40 per cent; fall of potential 5 volts per cell; capacity of cells, 1,200 amps; capitalization, $50,000; 360 days per year; 24 hours per day:

*Costs Per Year.*

| | |
|---|---:|
| 650 tons Sodium hydroxide at $45.00 | $29,250 |
| 725 horse-power years at $20.00 | 14,500 |
| Thirty men at $2.00 per day per man | 21,600 |
| Management and expert labor | 9,000 |
| Interest and depreciation 15 per cent | 7,500 |
| General expenses and selling, 5 per cent of sale price | 9,000 |
| Miscellaneous supplies | 1,000 |
| Total | $91,850 |

Yearly production = 720,000 pounds.

Approximate cost per pound = $0.127.

As the market price of sodium shows no inclination to fall below $0.25 per pound, there seems to be ample margin for a handsome profit.

References:

    This Journal 1; 15 (1902).

    Zeit. f. El-ch. 8; 412 (1902).

    Eng. & Min. Jr. 57; 580 (1894).

    U. S. P. No. 452,030 of May 12, 1891. Also patents in Eng., Ger., Fr. and Switz.

The processes used by the Aluminium Industrie Aktengesellschaft and the Chemische Fabrik Griesheim Elektron, of Bitterfield, are somewhat similar, both making use of the surface contact electrode for the cathode. The Bitterfield process was devised by Rathenau and Suter and their apparatus is of very simple in construction. The caustic soda is fused in an iron vessel. The anodes dip into the electrolyte, but the cathodes just touch the surface where the sodium gathers in a globule. These inventors do not consider it necessary to use a diaphragm.
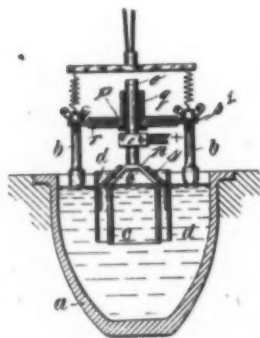


FIG. 6.—BOELSTERLI CELL.

The metal has to be gathered in even smaller dribblets than in the Castner process. Ger. P. No. 96,672 of March 15, 1896.

The Aluminium Industrie process was invented by J. Boelsterli. In his apparatus the metal is liberated exclusively at the surface of the electrolyte. The cathodes b (Fig. 6) have rounded ends, and are arranged in a ring around the anode, or cathodes and anodes may be arranged alternately. The anode c dips some distance below the surface of the electrolyte and is surrounded by a sheath d, which retains the oxygen and causes it to pass to the surface within the space which it encloses. This process has the same disadvantage as the Bitterfield process with regard to the collection of the metal in small amounts,

and has the further drawback that the solid shield between the electrodes must greatly increase the required voltage. U. S. P. No. 589,523 of Sept. 7, 1897. Also patents in Ger., Eng. and Switz.

An apparatus described by Dronier may have been intended for the electrolysis of fused sodium hydroxide, but as both electrodes dip deeply into the electrolyte and have no diaphragm between them, it does not seem to be very well adapted to the purpose.

Reference:

    Zeit. f. El-ch. 4; 419 (1898).

George P. Scholl, of Philadelphia, Pa., proposed to reduce the voltage required for the preparation of sodium by adding sodium sulphide to the bath. He states that the sulphur liberated at the anode unites with the sodium hydroxide to form sodium sulphide, and thus by adding fresh quantities of the hydroxide as required, it is possible to have the continuous formation of sodium at the low voltage of decomposition of sodium sulphide. The patent specifications say that less hydrogen is liberated than in the Castner process, but it does not state what becomes of the hydrogen in the hydroxide that must be constantly added. The writer is inclined to believe that the sulphide would be oxidized to sulphate at the anode, and that this sulphate would diffuse in the bath and be reduced at the cathode, thus depolarizing at both electrodes and reducing the ampere efficiency. The presence of the sulphate in the bath would also raise the melting point until a point would finally be reached where the temperature would be so high that the preservation of the liberated sodium would be a matter of considerable difficulty. U. S. P. No. 679,997 of Aug. 6, 1901.

To raise the low ampere efficiency of this class of processes, Thomas Ewan, assignor to the Cassel Gold Extracting Co., suggested the use of a porous diaphragm to separate the containing vessel into anode and cathode compartments. The diaphragm is made of alumina or sodium aluminate. The water formed in the anode compartment is removed by passing air through the electrolyte. The difficulty lies in the doubtful durability of the diaphragms in the fused bath. (See Darling process under next class for fuller discussion of this difficulty.) In the "Analysis of Current Patents," this journal, 2; 70 (1904), some comments were made by the writer of the analysis which do not seem quite justified by the facts. The comments were as follows: "He omits, however, in propounding this novel theory of the electrolysis of fused caustic soda, to state where the water, which he claims is formed at the anode, is to come from. Blowing air through or over the caustic melt would result in the formation of a good deal of carbonate with consequent thickening of the melt." The first portion of these objections has been met by the theory of the process as explained in this article, and the second is not of such vital importance. It has been shown by the writer (this journal 2; 358 [1904]) that the presence of about 40 per cent of sodium carbonate does not raise the melting point of sodium hydroxide. This carbonate will not be decomposed by the current, but some time would be required to convert 40 per cent of the bath to carbonate. As the above-mentioned research was not published until after the "Analysis" was written, its writer cannot be held responsible for facts of which he had no means of knowing. U. S. P. No. 745,958 of Dec. 1, 1903. Also patents in Eng., Ger. and Fr.

### Electrolysis of Fused Sodium Nitrate, 669.733.374.5.

Some experiments on the electrolysis of fused sodium nitrate were made by Faraday in 1833 and Hittorf in 1847, but their results are of no importance from the technical standpoint.

References:

    Faraday, Philosoph. Trans. (1833).

    Ostwald's "Klassiker," No. 86; 44.

    Hittorf, Pogg. Ann. d. Phys. 72; 481 (1847).

The Darling process seems to be the only attempt on record toward the technical solution of this problem. In many ways

it is a very attractive proposition. The melting point of the nitrate is low; the raw material is not much more expensive than sodium hydroxide, and nitric acid, a valuable by-product, is obtained, a by-product so valuable in fact that in a normally efficient process it would pay the cost of the raw material and the power consumed; the theoretical ampere efficiency is high, and the properties in general would seem to lend themselves to treatment by a fusion process. In spite of these advantages, however, the technical difficulties are even greater than in the electrolysis of fused salt. The electrolyte is very corrosive and is a powerful oxidizing agent, while the metallic sodium is an equally efficient reducing agent. It is, therefore, quite evident that metallic sodium cannot be liberated in a bath of fused sodium nitrate, and that to accomplish it some form of diaphragm must be used.

The Darling cell consisted essentially of a cylindrical iron melting vessel within which was a large cylindrical porous cup of special construction. This cup was made up of two concentric, perforated sheet steel shells, the space between which was filled with magnesia fritted in the electric furnace. Later, a mixture of magnesia and Portland cement was used for the filling. The cathode compartment within the cup contained fused sodium hydroxide. The melting vessel itself acted as anode, and the fused nitrate was contained in the space between the melting vessel and the porous cup. The nitrous fumes formed at the anode were conducted away and absorbed in water, to form nitric acid. In order to protect the metal walls of the diaphragm they were placed in shunt with the anode, thus making them cathodes. In order to prevent the migration of the nitrate into the cathode compartment it was necessary to thicken the bath by adding a little alumina. This cell produced sodium with a fairly good efficiency, but the life of the diaphragm is only 400-450 hours, making very heavy charges for maintenance and attendance. A further disadvantage was that the fall of potential was about 15 volts per cell. Experiments were made on a large scale and large quantities of sodium were produced, but for the reasons given above, it did not prove to be a commercial success. This lack of success seems to have deterred others from making any similar experiments, and the Darling process stands as the one attempt toward the technical solution of this problem.

References:

Jo. Frank. Inst. 153; 65 (1902).
El. W. & Eng. 39; 136 (1902).
Zeit. f. El-ch. 9; 369 (1903).
U. S. P. No. 517,001 of March 20, 1894.
U. S. P. No. 590,826 of Sept. 28, 1897.
U. S. P. No. 641,276 of Jan. 16, 1900.
U. S. P. No. 641,438 of Jan. 16, 1900.
Also patents in Eng., Ger., Fr. and Switz.

## COMPOUND ELECTROLYSIS, 669.733.374.6.

This class includes processes in which the electrolysis takes place in two steps instead of one, both steps proceeding simultaneously and involving the use of a liquid bipolar electrode, or its equivalent.

The best common example of a technical application of a liquid bipolar electrode is the Castner process for preparing sodium hydroxide. Its chief advantage over the other processes for the preparation of sodium hydroxide lies in the complete isolation of the anode and cathode products from each other. This is the most perfect method of preventing depolarization by the products liberated at the other electrode, and, if it works so well with solutions, why cannot the same principle be applied to fused baths? We have already seen that the direct electrolysis of fused sodium chloride brought forth nothing but failure from the technical standpoint, largely because it proved to be impracticable to liberate both chlorine and sodium in the same bath, even where there were partitions partially separating the two compartments from each other. If, now, a Castner cell is imagined in which fused sodium

chloride takes the place of the salt solution, melted lead is substituted for the mercury and the cathode in fused sodium hydroxide instead of solution of the same, it would seem feasible to produce metallic sodium instead of sodium hydroxide in the cathode compartment.

The germ of this idea is given in the basic Castner patent (U. S. P. No. 528,322 of Oct. 30, 1894), in which it says: "The same principle may be applied to the electrolysis of fused salts using a molten metal such as lead, tin or the like, or an alloy of various metals in place of mercury." It is possible that this inventor might have developed this idea himself had it not been for his untimely death.

A fused metal alloy as a soluble anode in a fused bath has been made use of in the refining of several metals. The refining of crude lead in a bath of fused lead chloride is described in Blount's "Practical Electrochemistry, p. 92, second edition. The same principle has been used by William Hoopes for the purification of aluminium. U. S. P. No. 673,364 of April 30, 1901.

The first attempt to apply this idea to the preparation of sodium was that of E. A. Ashcroft. His process has been recently described in this journal (4, 220 [1906]), and before the Ithaca meeting of the American Electrochemical Society (Transactions 9; 123 [1906]), so it is not necessary to repeat in full. The author, however, has done considerable work with a furnace of 1,000 amps. capacity, similar to Ashcroft's, and is therefore in a position to add something to the literature of the subject, pointing out from "commercial-scale" experience some of the advantages and disadvantages of this method.

The Ashcroft process, as described in the above-mentioned reference consists in electrolyzing a fused bath of sodium chloride with a fluid lead cathode and carbon anodes. The lead alloy is then circulated through an "equalizer," where it is cooled to about 350° C., into a second compartment, where it acts as a fused soluble anode in a bath of fused sodium hydroxide. The depleted alloy then returns through the equalizer, where it is warmed by cooling the hotter alloy fresh from the anode compartment. The lead is circulated by passing magnetic lines of force at right angles to the flow of the current.

In the apparatus used by the author, the lead was first circulated by pistons (U. S. P. No. 830,051 of Sept. 4, 1906), but later by means of a screw propeller. The alloy was circulated directly from one compartment to the other, therefore, the temperature in the cathode compartment was about as high as in the anode compartment. At that high temperature (about 800°), no sodium was obtained when fused sodium hydroxide was used as the electrolyte, but by using a mixture of the chlorides of sodium, potassium and calcium, sodium was recovered without any very great difficulty, the chief precaution necessary being the protection of the separated metal from the air. The construction of the cathode compartment presented some difficulties, but none which cannot be overcome. The current density at the cathode was about 215 amps. per square decimeter (2,000 amps. per square foot), and at the anode about 270 amps. per square decimeter (2,500 amps. per square foot). If the current density be allowed to exceed 300 amps. per square decimeter (2,800 amps. per square foot), using graphite anodes in a bath of fused chloride at 700-800° C., the rate of corrosion becomes very rapid. At 270 amps. per square decimeter the electrodes were used several days without even rounding the corners.

The cooling and heating of the lead by the regenerative principle, as used in the Ashcroft process, is the source of some loss of heat, but this factor is not sufficiently large to interfere materially in the efficiency of the process. Assuming that 100 pounds of lead must be circulated for every pound of sodium produced, and that the lead returning to the anode compartment has to be heated 100°, the cost of supplying the lost heat by electrical energy is 0.05 cent per pound of sodium. The two great disadvantages are the high voltage required and the excessive wear and tear on the apparatus.

The fall of potential has been given by Ashcroft as 9 volts per furnace, and this figure corresponds with that found by the author in his furnace. To produce 1 kg. sodium by the Castner process at 40 per cent ampere efficiency requires 2,920 amp-hours at 5 volts = 14.6 kw-hours. To produce the same quantity of metal by the Ashcroft process at an ampere efficiency of 90 per cent, requires 1,297 amp-hours at 9 volts = 11.67 kw-hours. Thus while the ampere efficiency of the Ashcroft process is about 125 per cent greater than that of the Castner process, the energy efficiency is but 25 per cent greater. Ashcroft stated in the above reference that the energy efficiency of the two processes is approximately the same, but the author is of the opinion that he overestimated the energy efficiency of the Castner process. The power factor is, however, of much less importance than the cost of the raw materials, and it is in the difference in the cost between sodium hydroxide and salt that the advantage of the Ashcroft process rests.

For high temperature furnaces, both the original cost and the wear and tear are higher than for the simple cells which can be used for the electrolysis of fused sodium hydroxide. The lead used for the cathode costs about $50.00 for every 1,000 amps. capacity, and must be included in the cost of the furnace. While, theoretically, no lead is lost, there is some loss in practice.

Mr. Ashcroft has estimated the cost per pound of sodium by the Castner process at 10.5 to 14.5 cents, and by his own process at 5 to 9 cents, according to the cost of power. From the data determined by the author, a yearly estimate of the costs has been made, under the following conditions: Production of 1 ton per day; ampere efficiency, 90 per cent; fall of potential, 8 volts per furnace; capacity of cells, 5,000 amps.; capitalization, $60,000; 360 days per year; 24 hours per day.

### Costs Per Year.

| | |
|---|---|
| 960 tons of salt at $5.00...................... | $4,800 |
| 600-hp. years at $20.00...................... | 12,000 |
| Thirty men at $2.00 per day per man......... | 21,600 |
| Management and expert labor................ | 9,000 |
| Interest and depreciation at 20 per cent....... | 12,000 |
| General expenses and selling at 5 per cent of sale ................................ | 9,000 |
| Miscellaneous supplies ...................... | 1,000 |
| | $69,400 |

Yearly production 720,000 pounds.
Approximate cost per pound = 9.63 cents.

From the above costs must be deducted any profit that can be realized from the utilization of the chlorine, but as the chlorine from fusion cells is not in as good condition for the production of bleaching powder as that which comes from solution cells, the profit from this source may be quite small. Under similar conditions, this makes an advantage of about 3 cents per pound in favor of the Ashcroft process. Whether the Castner process will be abandoned or not depends upon numerous other factors than those above discussed, but it seems hardly probable that any new plants will be erected using the Castner process even after the Castner patent expires.

References:
This Journal 4; 220 (1906).
U. S. P. No. 788,506 of May 2, 1905.
U. S. P. No. 801,199 of Oct. 10, 1905.
Also patents in Eng., Ger. and Fr.

### ELECTROLYSIS OF SOLUTIONS, 669.733.375.

There are two general ways in which metallic sodium can be prepared from sodium compounds in solution; first, by electrolyzing such a solution with a mercury cathode and distilling off the mercury from the amalgam so obtained, and, second, by using a solvent for the sodium compound which is to be electrolyzed, which will not react with sodium.

It is not within the province of this article to describe the numerous apparatus available for the preparation of sodium amalgam, even though every such a one has in it a possible preliminary step in the separation of metallic sodium. Numerous inventors have incidentally suggested the use of the amalgam for the preparation of sodium, but there seem to be only two inventors who have intended that the metal should be isolated continuously in this manner. Baker and Burwell proposed to produce sodium amalgam continuously in the conventional way. The amalgam was simultaneously distilled and the condensed mercury vapor returned to the mercury cell. This process has a high ampere efficiency, so far as the formation of the amalgam goes, but there is one serious if not fatal objection to the process. There are few, if any, mercury cells that produce sodium amalgam containing more than 0.1 per cent of sodium. It would, therefore, seem doubtful whether it would pay to distill off half a ton of mercury in order to obtain 1 pound of sodium.

References:
U. S. P. No. 734,499 of July 28, 1903.
U. S. P. No. 739,139 of Sept. 15, 1903.
U. S. P. No. 782,893 of Feb. 21, 1905.

The commercial prospects of the electrolytic deposition of sodium from organic solvents, or other solvents that will not react with sodium, do not seem very bright. An interesting paper on this subject was presented by Patten and Mott before the Cleveland meeting of the American Chemical Society, June 29 and 30, 1903. Extracts therefrom may be found in this journal 1; 419 (1903).

### SODIUM ALLOYS, 669,733.6.

The alloys of sodium have been used more as an intermediate product in the production of sodium hydroxide than in the preparation of metallic sodium, and the preparation of the amalgam especially is more closely related to the alkali industry than to the metallurgy of sodium, but since the preparation of lead-sodium alloy involves electrolysis in the fused state, it falls more naturally with the other fusion processes, and will, therefore, be treated here.

It is not possible to say positively who was the pioneer in the preparation of alloys of sodium with the heavy metals, but this honor is claimed by J. Walter, who states that he applied for a German patent on Aug. 24, 1887. The basic claim was on the preparation of sodium alloys by electrolysis using a fluid metal cathode with a constantly changing surface. In the *Zeitschrift für Elektrochemie* 3; 385 (1897), he gives a resumé of the original application, which shows such an advanced knowledge of the art that a full extract is given below:

"Neuerungen in der Herstellung von Metall-Legierungen auf elektolytischem Wege." On Aug. 24, 1887, the author applied for a German patent with the above title. The claim was on the preparation of alloys by electrolysis, using a fluid metal cathode with a constantly changing surface. This was denied, and German patent No. 42,418, cited. This patent is on the use of a moving cathode in electroplating. At the time this article was written, ten years after the above-mentioned application, several processes had been patented which would have been covered by this application. When one attempts to prepare sodium-lead alloy by electrolyzing a fused bath of some sodium salt with a fluid lead cathode, it is found that beyond a certain point the sodium is not absorbed by the lead as fast as liberated by the current, and escapes into the electrolyte. If the bath is allowed to cool it will be found that the top of the lead is very rich in sodium while the bottom is very nearly pure lead. The sodium-lead alloy is so much lighter than the lead that it mixes with difficulty. This is the great difficulty in the technical application of the process. This may be overcome by having the surface of the lead constantly renewed, which may be accomplished in various ways.

1. By mechanical stirring of the metal forming the cathode:
   a. By means of a neutral gas forced through the metal, or some vaporized neutral substance. This gas may be so chosen that it will depolarize the anode.

b. By means of mechanical stirrers, water or air cooled.

c. By turning the container or melting pot.

d. By raising a portion of the fluid cathode with an Archimedes screw or similar contrivance.

e. By an alternate flow of the cathode metal from one container to another.

f. By periodically adding fresh cathode metal.

2. By allowing the cathode metal to flow through the electrolyte by small falls or ridges or the like, the metal at all times, however, being connected with the cathode.

3. By hanging a solid bar of the metal into the melt. If the melting point of the metal is higher than that of the electrolyte and that of the alloy lower than that of the electrolyte, the alloy will drip off as formed, leaving a new surface of metal exposed.

4. By alternate enrichment of the bath near the cathode with one or more constituents in turn.

Examples of the application of this process:

a. To make sodium-lead alloy: Bath, sodium chloride; cathode, melted lead; anode, carbon. Cathode stirred by jet of nitrogen, hydrogen, carbon monoxide, methane, water gas, etc., which may also react with the chlorine and depolarize the anode.

b. Aluminium-zinc alloys: Bath, sodium and aluminium chlorides; cathode, a stream of zinc flowing over carbon.

c. Silicon-copper alloys: Bath, a fused mixture of alkali silicate and chloride. The cathode is a bar of copper. The temperature of the electrolyte is maintained at the melting point of the desired alloy. The alloy drips off the bar and collects in the bottom.

Whether this application was made in the form stated could not be confirmed by any literature available, but the author has assumed that the *Zeitschrift für Elektrochemie* would not have published the same had there not been some foundation in fact. It is a remarkable coincidence, however, that every process ever devised for the preparation of lead-sodium alloys is more or less completely covered by the above specifications. The author at any rate had a very clear perception of the difficulties involved and of the means for overcoming them.

The publication by Walter in 1897 was hardly as useful to the world at large as that of A. J. Rogers in 1889. The aim of his work was to produce sodium alloys to be used in the reduction of aluminium compounds. The "heavy metal" was placed in the bottom of a silicious crucible, which was heated in a wind furnace. By electrolysis of fused salt, using this heavy metal as cathode, he obtained alloys of lead and tin with sodium up to 50 per cent sodium with the tin and to 17 per cent sodium with the lead. In order to obtain pure sodium, these alloys must be distilled. The yield of sodium, as alloy, was 5-6 pounds sodium per horse-power per 24 hours. The crucibles are to some extent attacked by the alloys, but not so rapidly as by the pure alkali metals. A mixed electrolyte, composed of sodium chloride and cryolite, was also used. By applying the rich alloy thus obtained to the reduction of aluminium, the yield of aluminium was 1 pound per horse-power per 24 hours.

References:

Jr. Frank. Inst. 128; 486 (1889).

Proc. Wis. Nat. Hist. Soc. (1891).

Chem. News 60; 228 (1889).

The great year for investigation in this field was 1894, and the several workers of that period will be discussed in alphabetical sequence.

Borchers realized the necessity of keeping the lead cathode in motion, which he accomplished by allowing the fluid metal to flow down over a series of grooves arranged like small terraces, fresh lead being continuously added at the top and the alloy being continuously withdrawn from the bottom. The electrolyte was fused salt. U. S. P. No. 544,153 of Aug. 6, 1895.

The use of fused caustic soda as the electrolyte with a fluid metal cathode was suggested by Hetherington, Hurter and Muspratt. Unless the alloy obtained was to be used as an intermediary product in the preparation of the free metal, there is no use for the alloy thus formed. The author made some qualitative tests, using a fluid lead cathode in the electrolysis of melted caustic soda, and found that hydrogen as well as sodium was liberated at the cathode. Eng. P. No. 5,831 of March 20, 1894.

The lead-sodium alloy collects on the surface of the lead, and if allowed to remain will finally reach a degree of concentration at which the sodium will be redissolved as fast as liberated by the current. To overcome this, Hulin proposed to continuously deposit a certain proportion of lead with the sodium. This was accomplished by having a soluble lead anode, as well as insoluble anodes, in a bath of fused sodium and lead chlorides. A portion of the current was allowed to pass through the lead anode, the quantity of lead dissolved being proportional to the number of amperes allowed to pass through. The general construction may be readily understood from the illustration (Fig. 7), and no special description is necessary. In the large experimental plant the following results were obtained: Fall of potential, 7 volts per furnace;
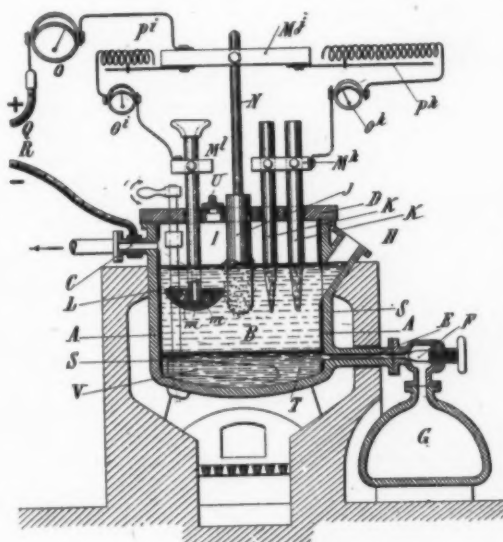


FIG. 7.—HULIN CELL.

current density at cathode, 70 amps. per square decimeter (650 amps. per square foot); yield, 1.24 kg. sodium per horse-power day (23 hours); ampere efficiency, 69.3 per cent; energy efficiency, 41.5 per cent. The low efficiencies said to be compensated for by the saving of evaporation charges and by the value of the by-product, lead peroxide which is obtained by heating the alloy to a low red-heat. The sodium is converted to oxide and then to hydroxide by treating with water.

References:

The Electrician 40; 623 (1898).

Eng. & Min. Jr. 67; 497 (1899).

U. S. P. No. 542,057 of July 2, 1895.

U. S. P. No. 587,830 of Aug. 10, 1897.

Also patents in Eng., Ger., Fr., Belg. and Switz.

The apparatus of Vautin at first consisted of a metal vessel lined with non-conducting material down to the surface of the fluid metal cathode. The alloy was drawn off intermittently. A later modification had a second vessel attached to the alloy-producing cell by means of a pipe. In this second cell the alloy was treated with steam to form caustic soda. No special means were employed for circulating the fluid metal cathode, an omission which was probably fatal to the commercial success of the process.

References:

    Industries & Iron 16; 386 (1894).
    Zeit. f. El-ch. 1; 250 (1895).
    Jr. Soc. Chem. Ind. (1894); 448.
    Zeit. f. angw. Chem. (1894); 528.
    U. S. P. No. 541,465 of June 25, 1895.
    U. S. P. No. 531,235 of Dec. 18, 1894.
    Also patents in Eng., Ger. and Switz.

In 1899, Haag sought to obviate the re-solution of the sodium in the fused bath by placing a layer of glass wool, asbestos or similar material on the surface of the fluid metal cathode.

References:

    Ger. P. No. 125,004 of April 6, 1899.
    Ger. P. No. 125,337 of April 6, 1899.

The largest technical application of lead-sodium alloys is the Acker process for making caustic soda. This consists briefly in the electrolysis of a fused bath of sodium chloride between carbon anodes and a fluid lead cathode. The lead-sodium alloy thus formed is forced into a second compartment by means of a steam jet, which simultaneously converts the sodium to hydroxide. The lead and caustic soda separate from each other by gravity in the second compartment, and the lead returns continuously to the electrolytic compartment. This process has been described so fully in this journal 1; 54 (1902), and in the Transactions of the American Electrochemical Society 1; 165 (1902), that it is unnecessary to repeat in full here, simply noting in passing that the name "Acker" is as imperishably associated with the preparation of lead-sodium alloy as is that of Castner with the direct electrolysis of fused caustic soda.

References:

    Zeit. f. El-ch. 9; 364 (1903).
    El. W. & Eng. 39; 586 (1902).
    Jr. Frank. Inst. 156; 221 (1903).
    Jr. Soc. Chem. Ind. 19; 47 (1900).
    U. S. P. No. 623,691-92-93 of April 25, 1899.
    U. S. P. No. 649,565 of May 15, 1900.
    U. S. P. No. 674,691 of May 21, 1901.
    U. S. P. No. 687,709 of Dec. 3, 1901.
    Also patents in Eng., Ger. Fr. and Switz.

The most recent attempt to utilize sodium-lead alloy in the preparation of caustic soda was that of H. S. Blackmore. For an adequate description of the process and apparatus, the original patent must be consulted. The surface of the lead is protected by a diaphragm composed of granular magnetite. The alloy diffuses to a second vessel with the aid of gravity, said vessel surrounding the electrolytic compartment, where the sodium is converted to caustic soda by the action of fused caustic containing a small quantity of water.

References:

    U. S. P. No. 759,799 of May 10, 1904.
    U. S. P. No. 809,085 of Jan. 2, 1906.
    U. S. P. No. 809,088 of Jan. 2, 1906

### CONCLUSION.

All the attempts to produce sodium commercially have failed except those involving the direct electrolysis of fused caustic soda. The only important use of the lead-sodium alloy is in the production of caustic soda. For further improvement the only hope seems to be in the electrolysis of fused salt, the field where the greatest number of failures have been recorded. It is quite probable that a successful solution of the problem has been found in the Ashcroft compound process. At the present price of sodium it seems probable that its use will still be confined to the preparation of sodium cyanide, sodium peroxide and a few minor applications, but it is to be hoped that the price may be further reduced and new uses found. Perfection has not yet been attained, and the author hopes that this brief and fragmentary history may assist future workers in the improvement of the art.

## Mixed Distillation and Combustion Gases.

By Oskar Nagel, Ph. D.

If we subject natural uncarbonized fuel in proper apparatus (gas generators, also called gas producers) to incomplete combustion, mixed distillation and combustion gases are formed. In the upper layers of the producer the hydroscopic water is removed. In further going downwards the fuel (material to be gasified) is subjected to dry distillation, coke being the result of this process. The coke is burned incompletely in the lowest part of the producer, whereby, besides the heat necessary for evaporation and dry distillation, also CO is generated. The water which is introduced as moisture with the atmospheric air is also decomposed. A clear idea of these processes is given in the table below, without, however, taking into account the formation of tar, which is inconsiderable.

Composition of the coal used (bituminous coal of Ostran (Moravia) mixed with lignite of Leoben (Styria).

$$
\begin{aligned}
C &= 64.92 \\
H_2 &= 2.50 \\
N &= 0.50 \\
\text{Chemically combined water} \quad & 14.22 \\
\text{Hygroscopic water} \quad & 12.42 \\
\text{Ash} \quad & 5\ 44 \\
\hline
& 100.00 \\
\text{Combustible sulphur} \quad & 0.52 \\
\text{Calorific value} \quad & 6374 \text{ Calories.}
\end{aligned}
$$

(a) Process in the upper part of the generator (drying of coal): 100 kg. coal yield 12.42 water (steam), and 87.58 kg. dry coal.

(b) Process in the middle part of the generator (dry distillation of coal).

We suppose that the coke contains nothing but carbon, besides the ash, and that the gases of dry distillation contain no oxygen except as CO and $H_2O$ (the latter supposition is not quite but sufficiently correct, since the gases contain $CO_1$ and other oxygen compounds). The formation of tar is not taken into consideration.

| 87.58 Kg. dry coal contain | Coke Kg. | Gases of Distillation Kg. | | | | | |
|---|---|---|---|---|---|---|---|
| | | $H_2O.$ | CO. | $CH_4.$ | $H_2.$ | $NH_3.$ | $H_2S.$ |
| Ash..... | 4.92 | 4.92 | ...... | ...... | ...... | ...... | ...... | ...... |
| C....... | 64.92 | 58.73 | ...... | 5.67 | 0.52 | ...... | ...... | ...... |
| N....... | 0.50 | ...... | ...... | ...... | ...... | ...... | 0.50 | ...... |
| S....... | 0.52 | 0.12 | ...... | ...... | ...... | ...... | ...... | 0.40 |
| $H_2$..... | 4.08 | ...... | 0.635 | ...... | 0.17 | 3.14 | 0.11 | 0.025 |
| O....... | 12.64 | ...... | 5.08 | 7.56 | ...... | ...... | ...... | ...... |
| Sum..... | 87.58 | 63.77 | 5.715 | 13.23 | 0.69 | 3.14 | 0.61 | 0.425 |

Since only a small amount of N is present, we calculate the entire amount as $NH_3$; actually, however, but one-fifth of the nitrogen of coal is transformed into $NH_3$.

(c) Process on and just above the grate (incomplete combustion of the coke formed).

The coal analysis shows 5.44 per cent ash, while the table shows only 4.92 per cent, which is explained by oxidation, mainly formation of sulphates from $Fe_2S$. The composition of gas shown in the last table results from the average composition of generator gas and the composition of the gases of distillation, which is given above.

The distribution of heat in the generator is shown in the heat balance, Table I.

It is understood that the composition of generator gas de-

| Components in Kg. | Coke. | Air. | Sum. | Losses through Grate Openings. | Yields. | | | |
|---|---|---|---|---|---|---|---|---|
| | | | | | CO₂. | CO. | H₂O. | N. |
| Ash.... | 4.92 | ...... | 4.92 | 4.92 | ...... | ...... | ...... | ...... |
| C....... | 58.73 | ...... | 58.73 | 15.67 | 6.57 | 36.49 | ...... | ...... |
| N....... | ...... | 211.63 | 211.63 | ...... | ...... | ...... | ...... | 211.63 |
| S....... | 0.12 | ...... | 0.12 | 0.12 | ...... | ...... | ...... | ...... |
| H₂O {H₂ | ...... | 0.25 | 0.25 | ...... | ...... | ...... | 0.25 | ...... |
| {O | ...... | 64.49 | 64.49 | 0.25 | 17.51 | 48.65 | ...... | ...... |
| Sum..... | 63.77 | 276.37 | 340.14 | 20.96 | 24.08 | 85.14 | 0.25 | 211.63 |

pends, besides the quality of fuel, on the size of same, height of fuel layer, construction of generator, and also temperature and air pressure during the operation. Table I. was prepared by Richard Akerman.

### TABLE I.—HEAT BALANCE.

| PRODUCTION OF HEAT AND NON-PRODUCED HEAT. | SINGLE. | | COMBINED. | |
|---|---|---|---|---|
| | Cal. | % | Cal. | % |
| I. Production of heat: | | | | |
| 1. Heat produced in generator by chemical processes.............. | 179666.4 | 26.67 | | |
| 2. Heat introduced by coal and air (by their temperature)................. | 3337.9 | 0.49 | 183004.3 | 27.16 |
| II. Non-produced heat: | | | | |
| 1. Unburned coal falling through the grate............................ | 126613.6 | 18.79 | | |
| 2. Heat capacity of generator gases ... | 364028.0 | 54.05 | 490641.6 | 72.84 |
| III. Heat losses: | | | | |
| 1. By fuel and ash falling through grate | 2316.1 | 0.34 | | |
| 2. By heat carried away by the gas produced.......................... | 28282.0 | 4.20 | | |
| 3. Loss by moisture of gas............ | 12346.3 | 1.83 | | |
| 4. By decomposition of water.......... | 8615.5 | 1.28 | | |
| 5. (a) Radiation...................... | 94890.7 | 14.09 | | |
|    (b) Heat necessary for gasifying coal | 36553.5 | 5.42 | 183004.3 | 27.16 |
| IV. Non-produced heat: | | | | |
| By unburned coal falling through grate | | | 126613.6 | 18.79 |
| | | | 309617.9 | 45.95 |
| Heat gained................. | | | 364028.0 | 54.05 |
| | | | 673645.9 | 100.00 |

### A—GENERATOR GAS FROM WOOD OF FIR TREES.

| KIND OF FUEL. | Trunks and Roots of Fir Tree. | Brush-wood of Fir Tree. | Logwood of Fir Tree. | Sawmill Refuse. |
|---|---|---|---|---|
| Size... {Thickness...............mm | 20-35 | ...... maximum 200 | 35-150 maximum 890 | 20-200 maximum 340 |
| {Length................mm | 500-750 | | | |
| Contents: | | | | |
| Hygroscopic water, %.......... | 12. | 16. | 27. | 60. |
| Ash, %...................... | 0.9 | 0.6 | 0.5 | 0.3 |
| Wood substance, %........... | 87.1 | 83.4 | 72.5 | 39.7 |
| Composition of wood substance. | | | | |
| C %....................... | 53.0 | ? | 51.0 | ? |
| H₂ %...................... | 7.1 | ? | 6.1 | ? |
| O %....................... | 39.8 | ? | 29.4 | ? |
| N %....................... | 0.1 | ? | | ? |
| Grate area, square meter, of gen... | 0.0 | 0.81 | 1.72 | 1.37 |
| Cubic content, cubic meters, of gen. | 26.7 | 1.9 | 24.2 | 7.4 |
| Consumption of fuel per day: | | | | |
| Per sq. meter grate area {m³.... | ...... | 8.1 | 23.8 | 14.4 |
| {kg.... | ...... | 1654 | 8891 | 7909 |
| Per generator.......... {m³..... | 65.2 | 6.6 | 41.0 | 19.7 |
| {kg..... | 14866 | 1340 | 15293 | 10835 |
| Number of charges per 24 hours.. | 2.8 | 5.6 | 4.1 | 6.6 |
| Length of time of presence of fuel in generator (hours)............... | 8.6 | 4.3 | 5.9 | 3.6 |
| Temperature of gas leaving generator, degrees C............ | 180° | 505° | 147° | 125° |
| Kg. tar in 24 hours............. | ? | ? | 444 | ? |
| Composition of tar: | | | | |
| C %....................... | ...... | ...... | 75.5 | ...... |
| H₂....................... | ...... | ...... | 7.4 | ...... |
| O....................... | ...... | ...... | 16.6 | ...... |
| N....................... | ...... | ...... | 0.5 | ...... |
| Volume composition of gases free of moisture and air: | | | | |
| CO₂....................... | 3.8 | 6.2 | 6.0 | 11.3 |
| CO....................... | 29.8 | 26.0 | 29.8 | 19.6 |
| C₂H₄..................... | 0.6 | ...... | 0.3 | 0.9 |
| CH₄..................... | 4.2 | 5.1 | 6.9 | 4.3 |
| H₂...................... | 6.4 | 4.3 | 6.5 | 7.4 |
| N₂...................... | 55.2 | 58.4 | 50.5 | 56.5 |

### B—GENERATOR GAS FROM PEAT.

| Origin. Quantity of Peat. | Munkfors Good Fibrous Peat. | Lotorp Good Fibrous Peat. |
|---|---|---|
| Intermediate analysis {Hygroscopic water %............. | 25.0 | 36.0 |
| {Gases, noncombustible......... | 8.3 | 17.6 |
| {Gases, combustible............ | 39.0 | 16.9 |
| {Fixed carbon %............... | 24.9 | 24.0 |
| {Ash %....................... | 2.8 | 5.5 |
| Composition of peat substance {C % | 57.8 | 61.0 |
| {H₂ % | 6.8 | 6.3 |
| {O % | 34.0 | 30.6 |
| {N % | 1.4 | 2.1 |
| Grate area square meters..... {of generator | 0.0 | 1.6 |
| Cubic content, cubic meters... | 22.8 | 21.9 |
| Daily fuel consumpt'n {Per sq. meter grate area {Cubic meter | .... | 12.8 |
| {Kg........ | ...... | 5279 |
| {Per generator.......... {Cubic meter | 20.6 | 20.4 |
| {Kg........ | 6262 | 8446 |
| Number of charges per 24 hours......... | 1.3 | 1.1 |
| Length of time for which fuel remains in generator in hours................. | 18.5 | 21.8 |
| Temperature of gas leaving producer deg. C. | 86-100° | 75-105° |
| Kg. tar in 24 hours.................... | 152 | 173 |
| Composition of tar {C........ | 79.6 | 79.8 |
| {H₂....... | 9.3 | 9.2 |
| {O....... | 11.1 { | 9.6 |
| {N....... | | 1.4 |
| Composition of gases free of air and water {CO₂ Vol. % | 6.6 | 6.8 – 7.4 |
| {CO.... | 29.6 | 27.6 –26.2 |
| {C₂H₄.... | 0.7 | 0.4 – 0.4 |
| {CH₄.... | 4.0 | 3.75– 3.70 |
| {H₂..... | 5.3 | 12.3 –13.5 |
| {N₂..... | 53.8 | 49.15–48.8 |

### C—GENERATOR GAS FROM BITUMINOUS COAL.

| | | |
|---|---|---|
| Intermediate analysis {Hygroscopic water, %............. | | 7.6 |
| {Gases, uncombustible, %........ | | 9.1 |
| {Gases, combustible, %......... | | 13.6 |
| {Coked coal, %................ | | 64.6 |
| {Ash....................... | | 5.1 |
| Composition of coal substance {C % | | 79.0 |
| {H₂ % | | 5.9 |
| {O % | | 13.7 |
| {N % | | 1.4 |
| Limestone addition, %................. | | 3.4 |
| Residue in ash-pit {Weight in % of coal............ | | 12.1 |
| {Composition.. {C % | | 40.2 |
| {H₂ % | | 1.0 |
| {O₂+N₂ % | | 1.2 |
| {Ash. | | 57.6 |
| Grate area, square meters, of generator.............. | | 2.0 |
| Cubic content, cubic meters, of generator.......... | | 4.0 |
| Daily consumption of coal per {Sq. m. grate area {Cubic meter | | 1.7 |
| {Kg........ | | 1251 |
| {Generator...... {Cubic meter | | 3.4 |
| {Kg........ | | 2502 |
| Number of charges per 24 hours.................... | | 1.2 |
| Length of time for which fuel remains in generator........ | | 20 |
| Temperature of gas leaving generator, deg. C........ | | 500° |
| Composition of gases free of air and water {CO₂ Vol. %.... | | 1.8 |
| {CO.... | | 27.3 |
| {C₂H₄.... | | 0.4 |
| {CH₄.... | | 4.2 |
| {H₂..... | | 6.2 |
| {N₂..... | | 60.1 |

### D—GENERATOR GAS FROM LIGNITE.

Below are given results with a lignite generator:

| | | |
|---|---|---|
| Number of generators...... | 3 | |
| Grate area per generator... | 2.5 square meter. | |
| Duration of test............ | 12 hours 45 minutes. | |
| Coal charged............. | 3600 Kg. Leoben (Styria) coal. | |
| Composition of coal {C. | | 61.72% |
| {Volatile H₂.................. | | 1.85% |
| {N........................ | | 0.22% |
| {H₂O chemically combined..... | | 20.09 |
| {H₂O hygroscopic.............. | | 9.34 |
| {Ash...................... | | 6.78 |
| {Combustible S................ | | 0.37 |
| Calorific value................................. | | 5446 Cal. |

Losses through grate............................ 936.7 Kg.
Composition of losse$\begin{cases} C............................ & 73.97\% \\ s \{ Ash...................... & 26.06 \end{cases}$

| | | I. | II. | III. | IV. | Avg. |
|---|---|---|---|---|---|---|
| Composition of the dry generator gas. | $CO_2$ Vol. %...... | 5.3 | 5.4 | 4.2 | 4.4 | 4.64 |
| | $O_2$............... | 0.3 | 0.8 | 0.6 | 0.8 | 0.65 |
| | CO............... | 25.19 | 25.05 | 25.39 | 26.50 | 25.59 |
| | $CH_4$............. | 0.29 | 0.15 | 0.51 | 0.40 | 0.38 |
| | $H_2$.............. | 10.29 | 10.65 | 11.29 | 11.60 | 11.11 |
| | $N_2$.............. | 58.63 | 57.95 | 58.01 | 56.30 | 57.63 |

The quantity gasified per hour and square meter grate area is:

Logwood and sawdust mixed.................... 45–50 Kg.
Sawmill waste................................ 200–330 Kg.
Logwood...................................... 370 Kg.
Loose peat (bad quality)...................... 75–120 Kg.
Good fibrous peat............................ 200–250 Kg.
Lignite...................................... 40–50 Kg.
Bituminous coal.............................. 60–250 Kg.

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## The Influence of Nickel and Carbon on Iron.

### By G. B. Waterhouse.

#### (Concluded from page 453.)

#### MICROSCOPICAL.

Transverse sections have been made of the bars in the rolled and other conditions, and also of K, P and Q, which contained respectively 0.41, 0.92 and 1.24 per cent carbon after they had been through the following treatment. They were placed in a Sauveur platinum resistance furnace, which was then luted up. In 10 hours they had reached 800° C., and in nine more hours they were at 1,080° C. The voltage was then lowered, and in 24 hours from the commencement of the operation the furnace was at 680° C. For 22 hours the temperature very slowly fell to 650° C., and then the current was cut off. The steels were removed after another 22 hours, being then at 100° C. The furnace temperature was controlled by a Le Chatelier thermo-electric pyrometer.

*Rolled Steels.*—These consist of ferrite, pearlite, cementite and temper graphite. K, as may be seen, has a small interlocking structure, of ferrite and pearlite. The pearlite increases in amount with increasing carbon, until with O and P it constitutes the entire field. Q, micrograph 4, with 1.24 per cent carbon, has a matrix of granular pearlite with thin cell walls of cementite, some of which is inside the crystals. The next sample, R, has more cementite, and a coarser structure than Q. S and T are represented by micrograph 6, which shows the globular amorphous temper graphite; the ferrite surrounding it; pearlite, and excess of cementite.

*Steels Submitted to Treatment "A."*—The specimens all show a coarsening of structure. The lower members consist of ferrite and pearlite. N A, micrograph 10, is almost entirely pearlite, and O and P are altogether composed of it. Q again consists of pearlite and cementite. The upper members of the series R, S and T, when polished, show the temper graphite, as seen in micrographs 12 and 14; and when etched present the rather unique combination of ferrite, pearlite, cementite and temper graphite as seen in micrograph 13.

*Steels Submitted to Treatment "B."*—In the lower members of the series annealing has brought about a complete separation of the pearlite into ferrite and cementite, as may be seen in micrographs 15, 16 and 17. Sample Q still shows cementite and pearlite. The higher members, when polished, have a great deal of temper graphite, micrograph 19. After etching the matrix is seen to consist of ferrite and free cementite, as in the case of KA or LA.

*Special Treatment in Electric Furnace.*—Sample K, micrograph 20. This is similar to KA, but the much slower cooling has allowed more complete segregation of the cementite.

Sample P, micrograph 21. Here is seen well laminated pearlite, with an excess of cementite in thin meshes.

Sample Q, micrograph 22, matrix of pearlite, with thick cell walls of cementite, some of which is also inside the crystals.

A sample of O in the condition after treatment B was taken for electrolysis, the method being that recommended by Arnold and Read. By means of a resistance in the circuit the current was kept at 1 amp. for 94 hours, when the piece was removed. The carbide was mostly attached to the bar. When removed it appeared as a grey sludge with a silvery sheen. The sample weighed originally 178.265 grams, and lost 31.23 grams by the operation. On analysis the figures given in Table V. were obtained.

#### TABLE V.

*Analyses of Separated Carbides.*

| | Carbide from OB. | Carbide from QA. |
|---|---|---|
| Carbon per cent...................... | 6.52 | 10.07 |
| Nickel per cent...................... | 1.86 | 2.79 |
| Iron per cent........................ | 91.71 | 71.61 |
| Silicon per cent..................... | 0.05 | 0.41 |
| Total............................... | 100.14 | 84.88 |

The same treatment was tried on a sample of Q in the condition after treatment A. During the electrolysis there was a continuous liberation of gases from the piece, some of which were undoubtedly hydrocarbons, judging from their odor. The carbide appeared as a dark-colored amorphous powder, and had the analysis given in Table V. The result will be discussed under a later heading.

#### PYROMETRIC.

The temperatures in this section were all obtained with a Le Chatelier thermo-couple consisting of wires of platinum and platinum alloyed with 10 per cent iridium. A delicate dead-beat mirror-galvanometer made by Carpentier was used, and the deflections observed by means of a scale and lamp in the usual manner.

For calibration the boiling points of water (100° C.), naphthalene (218° C.), and sulphur (448° C.); and the freezing points of tin (232° C.), zinc (420° C.), aluminium (655° C.), and copper (1,084° C.) were used, plotting temperatures and the galvanometer readings as co-ordinates.

The critical points were determined by taking pieces of the steels 1¾ inches long, and drilling a 5/16-inch hole ⅝ of an inch deep. Into this a fire-clay couple cover just fitted, bringing the thermo-junction in the center of the piece. The arrangement is shown in Fig. 3. The piece was packed in well-ignited asbestos in a crucible, and the heatings carried out in a small Fletcher furnace.

FIG. 3.—ARRANGE-MENT OF RECALES-CENCE PIECE.

The results obtained on heating and cooling are given in Table VI.

#### TABLE VI.

*Recalescence Results.*

| Mark. | C. C. per Cent. | Gr. Per Cent. | Ni. Per Cent. | Heating. Ac2.3. Degrees C. | Heating. Ac1. Degrees C. | Cooling. Ar3.2. Degrees C. | Cooling. Ar1. Degrees C |
|---|---|---|---|---|---|---|---|
| KA | 0.41 | .... | 3.79 | 744 | 704–714 | 653–638 | 608–598 |
| LA | 0.51 | .... | 3.79 | 734 | 699–714 | 640 | 615–605 |
| MA | 0.63 | .... | 3.76 | ... | 702–717 | ... | 610–600 |
| NA | 0.79 | .... | 3.81 | ... | 703–718 | ... | 612–602 |
| PA | 0.92 | .... | 3.79 | ... | 691–706 | ... | 612–602 |
| OA | 0.97 | .... | 3.75 | ... | 701–716 | ... | 612–602 |
| QA | 1.24 | .... | 3.81 | ... | 688–698 | ... | 620–610 |
| RA | 1.21 | 0.31 | 3.82 | ... | 683–693 | ... | 621–611 |
| SA | 0.93 | 0.71 | 3.82 | ... | 700–710 | ... | 623–613 |
| TA | 0.91 | 0.91 | 3.79 | ... | 696–711 | ... | 627–606 |
| RB | 0.50 | 1.02 | 3.82 | ... | 696–711 | ... | 615–605 |
| SB | 0.21 | 1.43 | 3.82 | 727 | 697–712 | ... | 626–606 |

The heating and cooling curves of the first three members of the series are given in Fig. 4. They, especially that of M, are typical of the curves of the whole series.
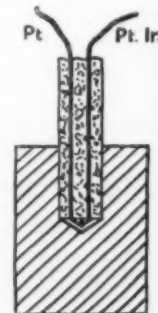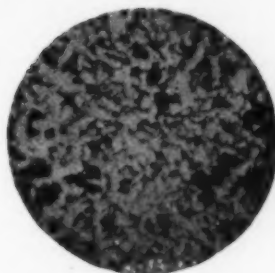
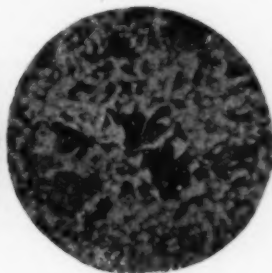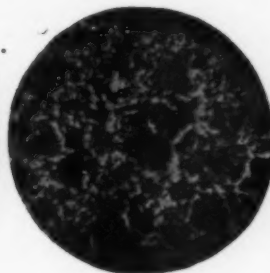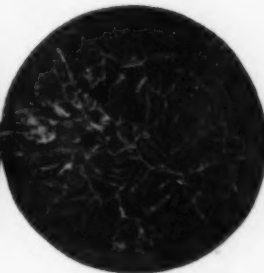On the advice of Prof. Howe, Fig. 5 was prepared, which

K No. 1.  Rolled.
169 diameters.

L No. 2.  Rolled.
169 diameters.

M No. 3.  Rolled.
169 diameters.

Q No. 4.  Rolled.
169 diameters.
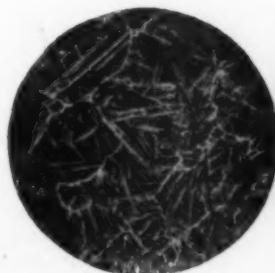
R No. 5.  Rolled.
169 diameters.
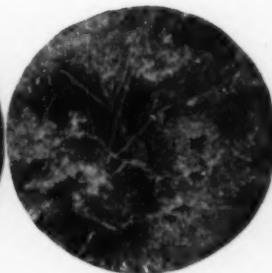
T No. 6.  Rolled.
255 diameters.
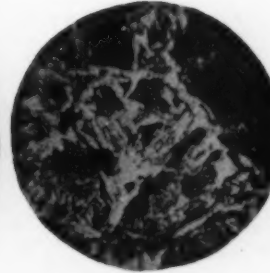
K No. 7.  Treatment A.
169 diameters.
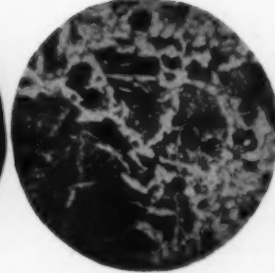
L  No. 8.  Treatment A.
169 diameters.
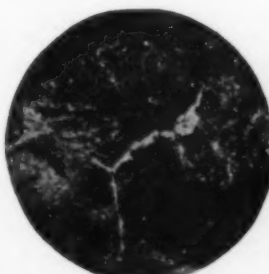
M No. 9.  Treatment A.
169 diameters.
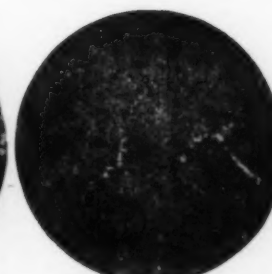
N  No. 10.  Treatment A.
169 diameters.

Q  No. 11.  Treatment A.
255 diameters.
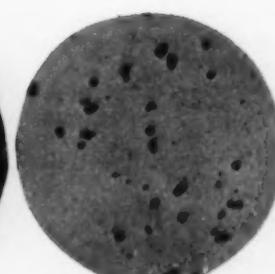
R  No. 12.  Treatment A.
199 diameters.  Polished.
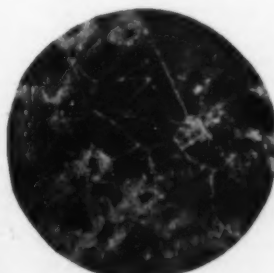
R  No. 13.  Treatment A.
255 diameters.
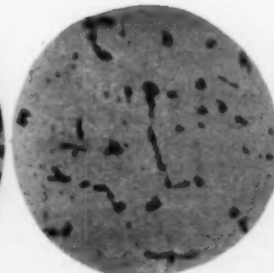
T No. 14.  Treatment A.
199 diameters.  Polished.

K No. 15.  Treatment B.
255 diameters.

M No. 16.  Treatment B.
255 diameters.

MICROPHOTOGRAPHS.

shows the relation of the critical points in cooling of these steels containing about 3.80 per cent nickel, to those of the pure carbon steels of the Roberts-Austen diagram.

Dr. Mathews examined the steels electrically. Comparing the electrical resistance with that of carbon steels of equal purity, it appeared that the addition of about 3.85 per cent nickel raised the specific resistance about 7.5 microhms per cubic centimeters in the steels submitted to treatment "B," for all percentages of combined carbon.
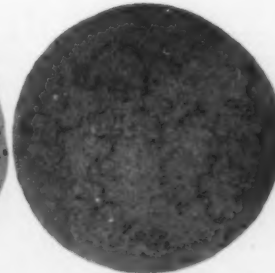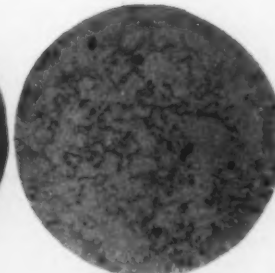
In the hardened state the increase in resistance was greater, and varied from 12 to 17 microhms per cubic centimeter. The electrical resistance of the rolled steels differed but little from those submitted to treatment "B," excepting in the event of



FIG. 4.—HEATING AND COOLING CURVES.

temper graphite being produced, 3.85 per cent nickel was almost without effect in increasing the coercive force of the steels, treatment "A," as compared with similar carbon steels in the same condition, but raised it considerably in the unannealed and hardened states.

Comparing samples K and R after treatment "B," when they contained nearly the same amount of combined carbon, they were found to have nearly the same coercive force and specific resistance, but R showed a lower hysteresis than K, and also lower tensile strength, elongation and reduction of area. The induction of sample K showed it to be the more permeable, but the difference was not great, and the general shape of the hysteresis loops was the same. In the rolled and treatment "B" steels, free from temper graphite, the hysteresis loss in ergs per cycle seemed to reach a maximum at sample Q. Treatment "B" reduced the coercive force in all cases, but the retentivity was often increased. The maximum induction for H-100 in this condition, was higher in some cases than in the rolled steels. This is not surprising when it is recalled that all bars were subjected to the same temperature, which might have been particularly good for some of them and bad for others. For H-25 to 30 the induction was increased by treatment "B" in every case, excepting sample O, in which it was greatly reduced. The hysteresis loss in ergs per cycle was determined in six of the specimens by Mr. Morecroft as follows:

| Number. | Rolled | Submitted to Treatment B. | Reduction. |
|---|---|---|---|
| K | 44,000 | 32,200 | 27.5% |
| M | 59,600 | 34,100 | 42.8% |
| O | 64,900 | 45,200 | 30.4% |
| Q | 65,700 | 47,000 | 28.5% |
| R | 64,000 | 25,700 | 60.0% |
| T | 64,200 | 21,200 | 67.0% |

*Consideration of Results.*—Examining the table of the mechanical tests and its accompanying diagram, it is seen that the action of carbon and nickel on iron is very similar to that

of carbon alone. In the steels after treatment "A," the tenacity, as shown by the elastic limit and ultimate strength, rises to a maximum with about 1.20 per cent carbon. Above this amount the tenacity drops in part, because some of the carbon is in the graphitic form.

The effect of the nickel is clearly shown by the great tenacity possessed by these steels as compared with carbon steel. To illustrate this the tests given by two similarly treated steels of Prof. Arnold's series may be given.

The elastic ratio, or the ratio of the elastic limit to the ultimate strength, is not greatly raised by the addition of the nickel, and, in the lowest member of the series, is slightly below that obtained from the carbon series. There is no doubt that in commercial nickel steels the elastic ratio is decidedly raised, which may be due to the large amount of manganese that they contain.

Corresponding to the increase in tenacity with the rise in carbon, there is a decrease in the ductility, as shown by the elongation and the reduction of area. At about 1.20 per cent carbon the ductility reaches a minimum, and then rises as the total carbon increases, which increase is accompanied by the separation of graphitic carbon.

The tensile properties are not greatly different from those of the pure carbon steels, and the influence of the nickel has been to increase the tenacity without lowering the ductility to any large extent.

The weakening effect of the separated temper graphite is well shown by the steels QA and RA, the tests of which are given in Table IV. They contain respectively 1.24 and 1.21 per cent of combined carbon.

The influence of treatment B has been very marked. The ultimate strength has been greatly lowered. It steadily rises with the increase in carbon until the latter is about 0.9 per cent, then rises slowly, attaining a maximum at about 1.2 per cent. The separation of the graphite carbon is marked by a drop in the curve.

The elastic limit of the steels after treatment B is singularly uniform. It was determined in all cases by an extensometer, which, on account of



FIG. 5.—COMPARISON OF CRITICAL POINTS WITH THE LOWER PART OF ROBERTS-AUSTEN'S DIAGRAM.

its superior sensitiveness, gave its indications before the drop of the beam, to which, however, it closely corresponded. This interesting point will be referred to later. The ductility of the steel has been increased by the treatment B. It does not greatly decrease after the carbon has reached 0.90 per cent, and shows a rapid increase with the appearance of the graphitic carbon.

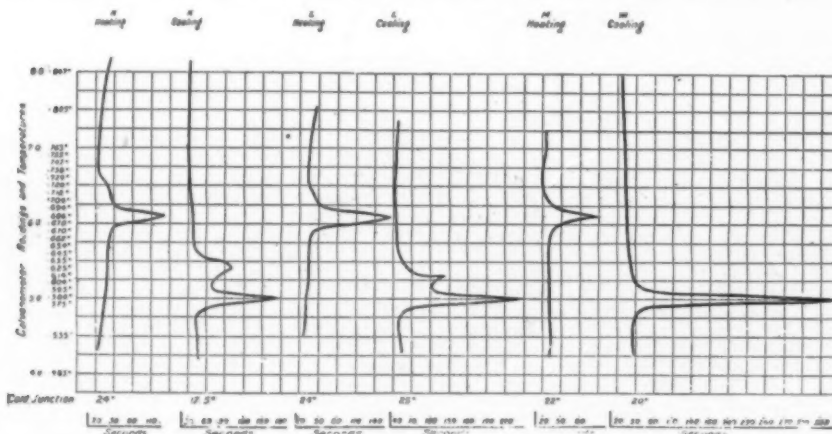In the case of Mr. Hadfield's nickel iron alloys, annealing

TABLE VII.
*Comparison with Carbon Steels.*

| Carbon. | Nickel. | Elastic Limit. | | Ultimate Strength. | | Elastic Ratio. | Elongation Per Cent. on 2 Inches. | Reduction of Area Per Cent. |
|---|---|---|---|---|---|---|---|---|
| | | Tons. | Lbs. | Tons. | Lbs. | | | |
| 0.38 | .... | 17.95 | 40,208 | 29.94 | 67,055 | 60.2 | 34.5 | 56.3 |
| 0.41 | 3.79 | 21.68 | 48,563 | 40.28 | 90,246 | 53.7 | 26.0 | 44.7 |
| 1.20 | .... | 35.72 | 80,012 | 61.65 | 138,096 | 57.8 | 8.0 | 7.8 |
| 1.24 | 3.81 | 45.44 | 102,030 | 75.31 | 168,697 | 60.3 | 3.5 | 3.5 |

did not have much effect, and he says: "The annealed cast bending bars, forged bending bars, and forged tensile bars show very little difference to the unannealed specimen° apparently showing that the annealing is of very little if any benefit." He expresses the wonder whether this would hold good with high carbon nickel steels, a question answered by these results in the negative.

The microscopical section shows that these steels belong to the first class of Osmond, and the pearlitic of Guillet, which is natural, because they lie wholly within the pearlitic area of his diagram. The constituents are those of carbon steels, namely, pearlite with ferrite as the excess substance in hypo-eutectoid or unsaturated, and cementite as the excess in the hyper-eutectoid or supersaturated steels. Graphite is also present. In the case of the rolled steels and those given treatment A, the free ferrite diminishes with increasing carbon, until P and O, containing 0.92 and 0.97 per cent of carbon, respectively, consist entirely of pearlite. Above this percentage cementite is found. In the steels which have been given treatment A the pearlite is of a sorbitic structure. In the higher members of the series the separated carbon has so lowered the percentage of combined carbon that the structure is mainly pearlite. The separated carbon is in small globules in the interior of the pearlite, and is surrounded by a layer of ferrite.

The annealing given in treatment B has had such an effect that the pearlite has separated into its constituents ferrite and cementite, and graphitic carbon is present in the higher members. In sample OB, containing 0.97 per cent carbon, the car-



O No. 17. Treatment B.
276 diameters.



Q No. 18. Treatment B.
183 diameters.



R No. 19. Treatment B.
285 diameters. Polished.

composition is above the eutectoid ratio; while the highest member, with 1.24 per cent carbon, has a large excess of cementite.

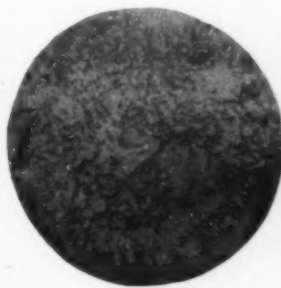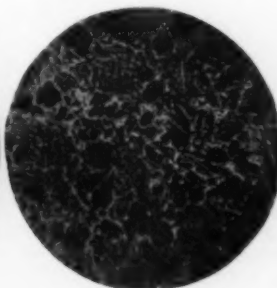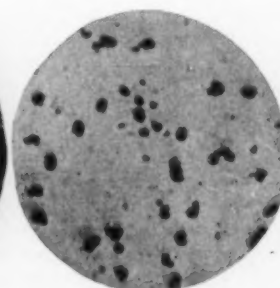In the steels given treatment B the cementite has the following composition: 6.25 per cent C., 1.86 Ni, 91.71 Fe. By dividing these percentages by the corresponding atomic weights we get 0.54 for C, 0.034 for Ni and 1.63 for Fe. The ratio of the carbon to the nickel and iron is 0.54 : 1.664, which equals 1 : 3.08, so that the formula of the cementite is $Fe(Ni)_{3.08}C$, or, writing it simply, $Fe(Ni)_3C$.

In the steels which have been given treatment A the finely divided, almost emulsified, cementite of the sorbitic pearlite suffers decomposition under the method employed for its separation, a moderate amount of hydrocarbons being given off, and free carbon also being produced. This makes it impossible to say what the analysis of the carbide is in the rolled steels and those given treatment A.

The results of the recalescence experiments show that the Ar1 point has been lowered 20° C. for each 1 per cent of nickel; and that the eutectoid ratio has been lowered by the nickel to about 0.70 per cent carbon. It is only in the steels that have been given the special treatment in the electric furnace that the eutectoid ratio appears to be about 0.70 per cent. In the rolled steels, and those subjected to treatment A, it appears to be about 0.95 per cent, while treatment B was so drastic that the pearlite was broken up into its constituents, and time allowed for them to segregate.



K No. 20. 215 diameters.



P No. 21. 374 diameters.

TREATMENT IN ELECTRIC FURNACE.
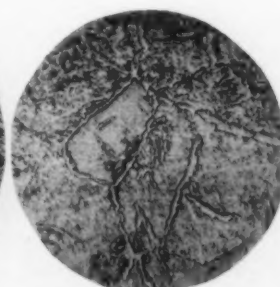


Q No. 22. 215 diameters.

bide contains 1.86 per cent nickel, and the ferrite has 1.89 per cent in solid solution. It would appear that the uniform elastic limit of these steels, and the fact that they are all chiefly ferrite and cementite, have a direct bearing on each other. Three specially annealed steels are particularly interesting. The lowest carbon steel has separated into ferrite and cementite; the one with 0.92 per cent carbon has well-developed pearlite with meshes of cementite, showing that this

CONCLUSIONS.

The following conclusions may be drawn from the results:

1. Nickel decidedly raises the tenacity without materially lowering the ductility. The elastic ratio, in pure nickel-carbon steels, is only slightly greater than that of carbon steels.

2. Annealing has a marked influence; it lowers the tenacity without greatly raising the ductility.

3. The constituents of steels with low percentage nickel in the unquenched state are: ferrite, pearlite, cementite and graphitic carbon.

4. The pearlite of these steels shows a great readiness to segregate into its constituents: ferrite and cementite.

5. In this condition the cementite has the formula $Fe(Ni)_3C$.

6. The eutectoid ratio in these steels appears to lie at about 0.70 per cent carbon, but in the rolled steels no free cementite shows until the carbon reaches about 0.95 per cent.

7. Nickel lowers the transformation points Ar3, Ar2 and Ar1 about 20° for every 1 per cent of nickel.

8. The cementite of these steels is very liable to precipitate its carbon as "temper graphite."

## Metallurgical Calculations.

### By J. W. Richards, Ph. D.

*Professor of Metallurgy in Lehigh University.*

### THERMO-CHEMISTRY OF THE BESSEMER PROCESS.

The feature of the Bessemer operation which strikes the observer as most wonderful, is that cold air is blown in great quantity through melted pig iron, and yet the iron is hotter at the end than at the beginning. If the observer will reflect a moment, however, he can see that if nothing but fuel, on fire, was in the converter, it would certainly be made much hotter by the air blast; in similar manner, the oxidation or combustion of part of the ingredients of the pig iron furnishes all the heat required for the process. Ten tons of pig iron contains, for example, some 350 kilograms of carbon, which is all burnt out in the bessemerizing, furnishing heat equal to the combustion of some 400 kilograms of coke—a not insignificant quantity, since it is burned and its heating power utilized in a very few minutes.

### Elements Consumed.

The usual ingredients of pig iron are:

| | | | | |
|---|---|---|---|---|
| Iron | 90 | to | 95 | per cent |
| Carbon | 2.5 | " | 4.5 | " |
| Manganese | 0.5 | " | 4.0 | " |
| Silicon | 0.5 | " | 4.0 | " |
| Phosphorus | 0.01 | " | 4.0 | " |
| Sulphur | 0.01 | " | 0.5 | " |

Some of the unusual constituents are nickel, chromium, titanium, aluminium, vanadium, tungsten, copper and zinc; all of these are rare, and there is seldom present as much as 0.5 per cent of any one, except in unusual cases.

In the Bessemer operation, carried out with the usual silica lining, iron, carbon, manganese and silicon are freely oxidized, but phosphorus and sulphur remain practically unchanged. In the basic Bessemer, lined with burnt dolomite and tar, phosphorus is also freely oxidized at the end of the operation, but sulphur is only slightly diminished—the more, the more manganese is in the slag. After all the oxidizable impurities are eliminated, iron itself oxidizes in much larger quantity, occasioning great loss if the blast is permitted to continue.

Iron oxidizes during the blow mostly to FeO, which enters the slag as ferrous silicate and partly to $Fe^2O^3$. The brown fume which escapes in large amount if the blow is continued too long, contains iron as $Fe^2O^3$.

| | | | | |
|---|---|---|---|---|
| Fe to FeO | 1173 | Calories per kg. of Fe |
| Fe to $Fe^2O^3$ | 1746 | " | " | " |
| Fe to $Fe^3O^4$ | 1612 | " | " | " |

The amount of iron oxidized can be gotten from the weight and percentage composition of the slag; also from the comparison of the weights of materials used and weight of ingots produced, knowing the weight of other impurities oxidized.

Carbon oxidizes mostly to CO gas, and partly, especially in the early part of the blow, to $CO^2$ gas. The proportion of each of these formed can only be known by analyzing the gases produced at various stages of the blow. The proportionate volumes of CO and $CO^2$ express the proportionate amounts of carbon forming each respective gas. A shallow bath allows more $CO^2$ to pass, on essentially the same principle that a deep layer of fuel on a grate favors the production of CO. The heat evolved by oxidation of carbon is:

| | | | | |
|---|---|---|---|---|
| C to CO | 2430 | Calories per kg. of C |
| C to $CO^2$ | 8100 | " | " | " |

Manganese oxidizes quickly and mostly to MnO. If the metal is a little overblown, $Mn^2O^3$ in small amount is found in the slag, while $Mn^2O^3$ is also present in the fumes. The heat evolved in these oxidations is:

| | | | | |
|---|---|---|---|---|
| Mn to MnO | 1653 | Calories per kg. of Mn |
| Mn to $Mn^2O^3$ | 2300(?) | " | " | " |

The last figure given is estimated; it has not yet been determined experimentally.

Silicon oxidizes rapidly and early in the blow to $SiO^2$, forming silicate slag with the metallic oxides formed. Its heat of oxidation has been usually taken as 7830 Calories per kilogram, but recent investigations have thrown doubt on this figure, Berthelot having found as low as 6414 Calories, and an American experimenter (research not completed; results not yet published) about 7000. Under these circumstances the best course is probably to use the middle value *ad interim*, and consider

Si to $SiO^2$ ...... 7000 Calories per kg. of Si

We hope that the exact figure will soon be determined.

Phosphorus oxidizes to $P^2O^5$, and only towards the end of the blow. It is practically completely eliminated, going into the slag as calcium phosphate.

P to $P^2O^5$ ........ 5892 Calories per kg. of P

Sulphur is not eliminated at all in the acid-lined converter. In the basic converter it is reduced in amount in the last few minutes, while phosphorus is disappearing, and partly escapes, mostly as $SO^2$ in the gases. The presence of a very basic slag is necessary, but the sulphur, while possibly going into the slag, does not remain there, but passes into the gases. The heat of oxidation of the unusual elements sometimes present are, as far as known,

| | | | | |
|---|---|---|---|---|
| Ni to NiO | 1051 | Calories per kg. of Ni |
| Ti to $TiO^2$ | 5000(?) | " | " | Ti |
| Al to $Al^2O^3$ | 7272 | " | " | Al |
| Zn to ZnO | 1305 | " | " | Zn |

The figure for titanium is estimated, and those for chromium, vanadium and tungsten are also unknown, although, as a guess and first approximation, might be used:

| | | | | |
|---|---|---|---|---|
| V to $V^2O^3$ | 2000(?) | Calories per kg. of V |
| W to $WO^2$ | 1000(?) | " | " | W |
| Cr to $Cr^2O^3$ | 3000(?) | " | " | Cr |

### Heat Balance Sheet.

Taking 0° C. (32° F.) as a base line, we may express the total heat contents of the pig iron, steel, gases, slag, blast, etc., from this temperature. This method is simpler than to take the bath at any one high temperature and to reckon from there. The items of heat available during the blow are:

Heat in the body of converter at starting.
Heat in the melted pig iron used.
Heat in the speigeleisen or ferro-manganese added.
Heat in hot lime added (sometimes in basic process).
Heat in the blast, if warm on entering.
Heat developed by oxidation of the bath.
Heat developed by formation of the slag.

The items of heat distribution are:

Heat in the body of the converter at finishing.
Heat in the steel poured.
Heat in the slag at finishing.
Heat in the gases escaping.
Heat in the fume.
Heat in the slag or metal blown out.
Heat absorbed in decomposing moisture of the blast.
Heat to separate the constituents of the bath.
Heat conducted away by supports, blast pipe, etc.

Heat conducted to the air in contact with converter.

Heat radiated during the blow.

These two columns should balance each other if all the items of each are correctly determined.

### Heat in Converter Body at Starting.

If the converter were quite cold when the pig iron was run in and the blow started, this item would be zero. But such a case would result disastrously, since the heat absorbed by the converter during the blow would be more than any ordinary heat could afford to lose. It is, therefore, customary, when starting for the first time, to build a fire in the converter and turn on a little air blast, so as to bring the inside up to bright redness, say 900° to 1000° C. The outside shell would, under these conditions, be at about 200°, and the mean temperature of the converter lining, say 400°. If the converter were in regular operation, one charge being introduced as soon as the other was finished, then the heat in the body of the converter at starting could be practically regarded as equal to the heat in the same at finishing, assuming the heats are running regularly. An estimate of the heat in the converter body at starting is therefore only necessary when the converter is first started up, or when it is allowed to stand some time between blows.

*Illustration:* Assume a converter weighing, without charge, 25 tons, of which 5 tons is iron work and 20 tons silica lining. The mean specific heat of iron (for low temperatures) being $0.11012 + 0.000025t + 0.0000000547t^2$, and for silica $0.1833 + 0.000077t$, calculate the heat contained in the body of the converter.

(1) When the temperature of the outside shell is 200° and that of the lining averages 400° (converter warmed up for starting).

(2) When the temperature of the outside shell is 300° and that of the lining averages 725° (converter empty at end of a blow).

*Solution:* (1)

Heat in 5000 kg. of iron work:

$0.11012 + 0.000025 (200) + 0.0000000547 (200)^2 = 0.11731$

$0.11731 \times 200 \times 5000 = 117,310$ Calories.

Heat in 20,000 kg. of silica lining:

$0.1833 + 0.000077 (400) = 0.2141$

$0.2141 \times 400 \times 20,000 = 1,712,800$ Calories.

Total heat in converter body = 1,830,110 "

It may be remarked that this would require the consumption of at least 250 kilograms of coke, with a calorific power of 8,000 Calories, to warm the converter to this extent.

(2) Heat in 5,000 kg. of iron work:

$0.12354 \times 300 \times 5,000 = 185,310$ Calories.

Heat in 20,000 kg. of silica lining:

$0.2391 \times 725 \times 20,000 = 3,466,950$ "

Total = 3,652,260 "

This condition is assumed as representing the converter when just emptied and immediately refilled. In this case, in regular working, the heat in the converter body is practically the same at the beginning and at the end of a blow, and the heat losses through it are only those due to conduction to the air and ground and radiation.

### Heat in Melted Pig Iron Used.

This quantity depends on the temperature at which the pig iron is run into the converter. If the iron is high in silicon, which would tend to produce a hot blow, it may be run in somewhat cool; but if low in silicon it should be run in much hotter. The minimum quantity of heat contained in a kilogram of melted pig iron may be put at 250 Calories; the maximum, for very hot pig iron, 350 Calories; about 300 Calories would be a usual average figure. This may be easily determined experimentally in any given case by granulating a sample in water in a rough calorimeter.

### Heat in Metallic Additions.

Melted spiegeleisen is usually not very hot when run into the converter. It may contain 250 to 300 Calories per kilogram; say an average of 275. Ferro-manganese is sometimes added red-hot, at 800° to 900° C. At this temperature it would contain 120 to 135 Calories per kilogram, assuming a mean specific heat of 0.15.

### Heat in Preheated Lime.

Taking the mean specific heat of CaO as $0.1715 + 0.00007t$, it would contain 154 Calories per kilogram at 700°, and 211 Calories at 900°. The heat content can be calculated for any known temperature at which the lime is used.

### Heat in Warm Blast.

No Bessemer converters are run by hot blast, but the air pressure used is so great (20 to 35 pounds per square inch) that the blast is warmed by compression in the cylinders. If air at 1 atmosphere tension (ordinary air) be compressed to 2 or to 3 atmospheres tension, giving effective blast pressures of 1 to 2 atmospheres, the air is heated 60° or 103°, respectively, above its initial temperature. While some of this heat may be lost in the cylinder and conduits, yet the air, unless artificially cooled, passes to the tuyeres at 25° to 50° C. above the outside temperature, and thus imparts some heat to the converter.

### Heat Developed by Oxidation.

We have already discussed the thermochemical data required for this calculation. To use the data we must find the weights of each ingredient oxidized (not forgetting the iron itself) and the nature of the oxide it forms. This is deduced from the analysis of slag, gases and steel produced, as compared with those of the pig iron and additions used.

### Heat of Formation of Slag.

The slag is a mechanical mixture or mutual solution of silicates of iron and manganese, with a little alumina (up to 5 per cent) and lime and magnesia from nothing up to 5 per cent. In the basic process a large amount of calcium phosphate is present, representing over half of the entire slag, while magnesia is present in considerable amount, and alumina is almost absent.

Having, in the preceding column, calculated the heat of oxidation of all the metals oxidized in the converter, it remains to calculate the heat of combination of these to form slag. The silicate of alumina contributes nothing, since it occurred combined in the lining or lime added. The same can be said of the lime and magnesia in the acid process slags. The FeO and MnO are then to be considered, and the only thermochemical data we have are:

(MnO, $SiO^2$) = 5,400 Calories.

(FeO, $SiO^2$) = 8,900 "

These data are for 71 or 72 parts of MnO or FeO respectively, uniting with 60 parts of $SiO^2$. Since in acid slags there is always proportionately less of the bases, we should utilize the above heats of formation by expressing them per unit weight of MnO or FeO going into combination. These figures are:

Per kg. of MnO = $5,400 \div 71 = 76$ Calories.

Per kg. of FeO = $8,900 \div 72 = 124$ "

From these figures the heat of formation of the slag can be calculated: Any $Fe^2O^3$ present in the slag would be calculated as its equivalent weight of FeO (by multiplying by $144 \div 160$).

*Illustration:* A Bessemer slag contained by analysis:

| | | |
|---|---|---|
| $SiO^2$ | 47.25 | per cent. |
| $Al^2O^3$ | 3.45 | " |
| FeO | 15.43 | " |
| MnO | 31.89 | " |
| CaO, MgO | 1.84 | " |

What is its heat of formation per kilogram?

*Solution:* The $Al^2O^3$ CaO and MgO are to be neglected,

for reasons already given. The heat of combination, per kilogram of slag, is therefore,

$$FeO \text{ uniting with } SiO^2 = 0.1543 \times 124 = 19.1 \text{ Cal.}$$
$$MnO \text{ uniting with } SiO^2 = 0.3189 \times 76 = 23.2 \text{ "}$$

$$\text{Total} = 42.3 \text{ "}$$

In basic Bessemer slags the conditions are much more complicated. The content of $P^2O^5$ is not under 14 per cent, runs as high as 25 per cent, and averages 19 per cent; the CaO averages 45 per cent, limits 35 to 55; the silica is usually below 12 per cent, and averages 6 to 8 per cent; magnesia is present from 1 to 7 per cent, average about 4 per cent. In such slags we should first of all assume the $P^2O^5$ to be combined with CaO as $3CaO, P^2O^5$, containing 168 of CaO to 142 of $P^2O^5$ (1.183 to 1), and having a heat of formation from CaO and $P^2O^5$ of 159,400 Calories per molecule, or 1123 Calories per unit weight of $P^2O^5$. Next the iron and manganese present may be calculated to FeO and MnO respectively, and treated as to their combination with $SiO^2$, the same as in an acid slag. Alumina may be considered in such basic slags as an acid, and equivalent to 120/102 of its weight of silica. These allowances will leave considerable lime and either an excess or deficiency of silica; if an excess, we can assume it combined with lime and magnesia, with a heat evolution equal to 476 Calories per kilogram of silica; if a deficiency, we let the calculations stand without further modification.

*Illustration:* Slag made at a Rhenish works contained:

| | | |
|---|---|---|
| $SiO^2$ | 7.73 | per cent. |
| $P^2O^5$ | 21.90 | " |
| $Al^2O^3$ | 3.72 | " |
| $Fe^2O^3$ | 1.00 | " |
| FeO | 4.73 | " |
| MnO | 2.05 | " |
| CaO | 50.76 | " |
| MgO | 4.00 | " |
| CaS | 1.71 | " |

What is its heat of formation per unit of slag?

*Solution:* The 21.90 parts of $P^2O^5$ would be combined with $21.90 \times 168/142 = 25.91$ parts of CaO. This leaves $50.76 - 25.91 = 24.85$ of CaO as either free, dissolved CaO or partly combined, in the slag. The 2.05 MnO would correspond to $2.05 \times 60/71 = 1.73 SiO^2$; the 4.73 FeO to $4.73 \times 60/72 = 3.94 SiO^2$; the 1.00 $Fe^2O^3$ to $1.00 \times 60/80 = 0.75 SiO^2$; a total $SiO^2$ requirement for these three bases of 6.42 per cent. The $SiO^2$ present is 7.73 per cent, adding to which the $SiO^2$ equivalent of the $Al^2O^3$ present ($3.72 \times 120/102 = 4.38$), we have 12.11 per cent of summated silica. The ratio of summated FeO to summated $SiO^2$ is considerably below the ratio 72 to 60, we can, therefore, consider the summated FeO as all combined with silica. The summated FeO is:

| | | |
|---|---|---|
| FeO | = 4.73 | per cent. |
| FeO equivalent of MnO = | 2.08 | " |
| FeO equivalent of $Fe^2O^3$ = | 0.90 | " |

$$\text{Total} = 7.71 \text{ "}$$

And the $SiO^2$ combining with this as $FeO . SiO^2$ is

$$7.71 \times 60/72 = 6.42 \text{ per cent.}$$

The excess of summated silica free to combine with lime is $12.11 - 6.42 = 5.69$ per cent. As there is 24.85 of CaO and 4.00 of MgO for it to combine with, the heat of this combination must be calculated on the $SiO^2$ going into this combination. We then have the formation heat of the slag as:

| | | |
|---|---|---|
| $P^2O^5$ to $3CaO . P^2O^5$....... | $21.90 \times 1123 =$ | 24,594 Cal. |
| MnO to $MnO . SiO^2$....... | $2.05 \times 76 =$ | 156 " |
| FeO to $FeO . SiO^2$......... | $5.63 \times 124 =$ | 698 " |
| $SiO^2$ to $3CaO . SiO^2$....... | $5.69 \times 476 =$ | 2,708 " |

28,156 "

This equals **281.6** Calories per unit weight of slag, forming

a very important item in the heat balance sheet, particularly when the slag is large in amount.

### Heat in Converter Body at Finishing.

This item reaches its maximum at the end of the blow, and would be equal to that calculated for the begining of the blow, with the exception that some of the lining, silica or dolomite has been corroded and passed into the slag, carrying with it its sensible heat.

### Heat in Finished Steel.

This should be determined experimentally in each particular case. If not so determined an average value may be assumed, based on the following considerations: The finishing temperature averages 1650° C.; at this heat average Bessemer steel will contain a total of 350 Calories of heat per kilogram. If the temperature is determined by a pyrometer, a correction of 1/5 Calorie can be made for every degree hotter or colder than 1650°.

### Heat in Slag.

Some experimental data are badly needed, concerning the heat in slags of different composition at different temperatures. At present it is necessary to make guesses, wherever the heat in the slag produced is not directly determined. At a finishing temperature of 1650° it is likely that the slag contains 550 Calories per kilogram; with a variation of 1/4 Calorie for each degree hotter or colder than 1650°.

### Heat in Escaping Gases.

The amount of these gases can only be determined satisfactorily from their analysis and the known weights of carbon oxidized. Direct estimation from the piston displacement is of much less exactness, because the slip and leakage at these high pressures may reach 25 to 50 per cent, or even more. The temperature of the gases is only slightly less than that of the bath, that is, some 1350° at starting and 1650° at finishing. Outside in the air the Bessemer flame may be much hotter than this, but that is due to further combustion of CO to $CO^2$ outside the converter, and should be disregarded. Where extra air is blown upon the surface of the bath, as in "baby" converters, it is quite possible that the gases in the converter and the escaping gases may be considerably hotter than the bath itself. These variations must be taken into consideration. The most satisfactory condition is to insert a pyrometer tube into the opening of the converter and measure the temperature directly. The heat carried out by the gases can then be calculated accurately, using the proper mean specific heats to these high temperatures already used in these calculations.

### Heat in Escaping Fume.

This is mostly oxides of iron and manganese, with sometimes silica. It is relatively small in amount. Its quantity being known, consider it at the same temperature as the gases, with a mean specific heat of 0.40 if free from silica, and 0.35 if siliceous.

### Heat in Slag or Metal Blown Out.

These can be counted as equal to the heat in an equal quantity of slag or metal at the finishing temperature.

### Heat Absorbed in Decomposing Moisture.

Knowing the amount of moisture blown in with the blast, a proper allowance is $29,040 \div 9 = 3,227$ Calories for every kilogram of moisture thus blown in. It is probable that this moisture is all decomposed, its hydrogen appearing in the gases. In the absence of data as to the hygrometric condition of the blast, the amount of moisture entering may be inferred and calculated from the amount of hydrogen in the gases.

### Heat to Separate Constituents of Bath.

We here meet the question, how much heat of combination exists between the bath and the various ingredients which are removed—carbon, silicon, manganese, phosphorus, sulphur. Le Chatelier believes manganese to exist in the bath as $Mn^2C$, requiring 80 Calories per kilogram of manganese to decompose

it. Silicon and carbon have, as far as has at present been determined, no sensible heat of combination with iron. Sulphur requires 750 Calories to separate each kilogram from iron. Phosphorus requires, according to Ponthiere, 1397 Calories to separate each kilogram from iron, but the reliability of this datum is doubtful. Until more reliable tests are made it is perhaps better to omit this item than to use it, although it must be of great importance if as large as Ponthiere states it to be.

### Heat Conducted Away by Supports.

This is a very difficult quantity to determine, being conditioned by the size of the supports, their cooling surface and the kind of connection they have with other objects. The heat which would pass to the blast pipe is practically returned to the converter by the incoming blast. The heat passing into the supports is perhaps best found by taking their temperature at different places, and calculating the heat loss from their surface by radiation and conduction to the air. This amounts practically to considering them as part of the outer cooling surface of the converter; the calculation of these surface losses is given in the two following paragraphs:

### Heat Conducted to the Air.

This is a function of the extent of outside surface, its temperature, the temperature of the air, and the velocity of the air current. Measurement will give the extent of surface in contact with the air and the average velocity of the air current; the surface being rough iron, the coefficient of transfer conductivity may be taken as $k = 0.000028 (2 + \sqrt{v})$, where v is the air velocity in c. m. per second, and k is the heat conducted per second in gram-calories, from each square c. m. of surface per $1°$ difference of temperature. The temperature of the outer surface should be carefully measured, so that a reliable average is obtained, and the air velocity likewise averaged, since it has considerable influence on the heat lost to the air.

*Illustration:* A converter has an outside surface of 50 square meters, at an average temperature during the blow of $200°$ C., the average air current being 1 meter per second, and outside air $30°$ C. What is the heat loss by conduction to the air in kilogram Calories per minute?

*Solution:*

Coefficient of transfer conductivity:

$$0.000028 (2 + \sqrt{100}) = 0.000336.$$

Heat loss per $1°$ difference, per second, in gram-calories:

$$50 \times 10,000 \times 0.000336 = 168 \text{ calories.}$$

Heat loss per $170°$ difference, per minute, in kg.-Calories:

$$168 \times 170 \times 60 \div 1000 = \textbf{1714} \text{ Calories.}$$

### Heat Radiated During the Blow.

This is a function of the temperature of the outside shell, the mean temperature of the surroundings of the converter and the nature of the metallic surface. As the surface is oxidized iron, it would lose about 0.0141 gram-calories from each square centimeter per second, if at a temperature of $100°$ and the surroundings at $0°$; or practically 1 gram-calorie per second from each square meter, for every 100,000,000 of numerical difference between the fourth powers of the absolute temperature of the radiating surface and its surroundings. (See Metallurgical Calculations, Part I., p. 185.)

*Illustration:* Assuming the surroundings of the converter at $30°$, in the preceding illustration, what amount of heat is radiated per minute in large Calories?

*Solution:* The absolute temperatures in question are $273 + 30 = 303$, and $273 + 200 = 473$. The difference of their fourth powers is:

$$473^4 - 303^4 = 41,626,500,000,$$

which, divided by 100,000,000 gives 416,265 gram-calories lost per second per each square meter of radiating surface.

The radiation loss for the whole surface per minute is, therefore:

$$416.265 \times 50 \times 60 \div 1000 = \textbf{1249} \text{ kg.-Calories.}$$

While the assumption made as to the temperature of the outside shell is doubtless only approximate, yet if the temperature of the same is carefully determined the radiation loss can be accurately calculated. If the outside of the converter were polished, this radiation loss might be reduced nearly nine-tenths.

### Problem 65.

From the data and results of calculation of Problem 62 (this journal, November, 1906,) we see that 22,500 pounds of pig iron and 2,500 pounds of spiegeleisen produced 24,665 pounds of steel, there being eliminated during the blow and recarburization:

Carbon ............... 679.5 pounds (140.7 to $CO_2$)
Silicon ............... 203.2   "
Manganese .......... 197.9   "
Iron ................. 253.9   "     (25.3 to $Fe_2O_3$)

The gases contain:

$CO_2$ ................... 5.20 per cent.
CO .................. 19.91   "
$H_2$ ................... 1.39   "
$N_2$ ................... 73.50   "

The slag contains:

$SiO_2$ ................ 63.56   "
$Al_2O_3$ ................ 3.01   "
FeO ................. 21.39   "
$Fe_2O_3$ ................ 2.63   "
MnO ................. 8.88   "
CaO ................. 0.90   "
MgO ................. 0.36   "

Make average assumptions for requisite data not given.

*Required:* A balance sheet of heat evolved and distributed.

*Solution:* The items of this balance sheet have already been discussed in detail. We will apply them to this specific case:

*Heat in Body of Converter at Starting:* Assuming that this is a blow in regular running, the heat may be taken at any reasonably approximate quantity, because the same quantity, with only a slight deduction, will be allowed as contained in the same on finishing. We will, therefore, take a figure already calculated, 8,034,970-pound Calories, as the heat in the converter body at starting.

*Heat in Melted Pig Iron:* We will take it at 300 Calories per pound, or a total of $300 \times 22,500 = 6,750,000$ Calories.

*Heat in Spiegeleisen:* $2,500 \times 300 = 750,000$ Calories.

*Heat in Blast:* This may safely be considered as warmed by compression and entering the converter at $60°$ C. The amount of blast received altogether is calculated thus:

Carbon oxidized             = 679.5 pounds.
Volume of CO and $CO_2$ formed:

$$679.5 \times 16 \div 0.54 = 20,133 \text{ cu. ft.}$$

Volume of gases $\dfrac{20,133}{0.2511}$             = 80,179   "

Volume of nitrogen 80,179 $\times$ 0.735  = 58,932   "
Volume of air proper in blast      = 74,408   "
Volume of hydrogen in gases:

                80,179 $\times$ 0.0139 = 1,115   "
Volume of moisture in gases        = 1,115   "

Assuming the blast at $60°$ C., the heat in it is:

Air     74,408 $\times$ 0.3046 =    22,665 oz. Cal. per $1°$
$H_2O$   1,115 $\times$ 0.3790 =       423   "     "

                        ——————
                        23,088   "     "

$$23,088 \times 60 = 1,385,280 \text{ oz. Cal.}$$
$$= 86,580 \text{ lb. Cal.}$$

*Heat of Oxidation:*

| | | |
|---|---|---|
| C to $CO^2$ | $140.7 \times 8100 = 1,139,670$ | Calories. |
| C to CO | $538.8 \times 2430 = 1,309,280$ | " |
| Si to $SiO^2$ | $203.2 \times 7000 = 1,422,400$ | " |
| Mn to MnO | $197.9 \times 1653 = 327,130$ | " |
| Fe to FeO | $228.6 \times 1173 = 268,150$ | " |
| Fe to $Fe^2O^3$ | $25.3 \times 1746 = 44,170$ | " |

$$4,510,800 \quad ''$$

*Heat of Formation of Slag:* The 197.9 pounds of manganese oxidized forms 255.5 of MnO. Hence the weight of the slag is $255.5 \div 0.0888 = 2877$ pounds. The slag, therefore, contains also $2877 \times 0.6356 = 1829$ pounds of $SiO^2$, of which $203.2 \times 60/28 = 435$ pounds came from the silicon oxidized, and 1794 pounds from the lining. The lining will also have lost the $Al^2O^3$, CaO and MgO in the slag, equal to $2877 \times 0.0427 = 123$ pounds. The total iron going into the slag, 253.9 pounds, is equivalent to 326.4 pounds of FeO.

The heat of formation of the slag will therefore be:

| | | |
|---|---|---|
| FeO | $326.4 \times 124 = 40,474$ | Calories. |
| MnO | $255.5 \times 76 = 19,418$ | " |

$$\text{Total} \quad 59,892 \quad ''$$

*Heat in Converter at Finishing:* This will be the same as at starting, less the heat in $1794 + 123 = 1917$ pounds of lining, which was corroded and entered the slag. Assuming this to have been on the inner surface, at an average temperature of 1500°, the heat in it, using the mean specific heat of silica, will have been

$$1917 \times 0.2988 \times 1500 = 851,200 \text{ Calories.}$$

And the heat in the converter body at the finish:

$$8,034,970 - 851,200 = 7,183,770\text{-pound Calories.}$$

*Heat in Finished Steel:* Taking its temperature as 1650°, with 350 Calories per unit, we have

$$24,665 \times 350 = 8,632,750 \text{ Calories.}$$

*Heat in the Slag:*

$$2877 \times 550 = 1,582,350 \quad ''$$

*Heat in Escaping Gases:* These have already been calculated as consisting of

| | | | |
|---|---|---|---|
| Nitrogen | 58,932 | cubic feet. | |
| Hydrogen | 1,115 | " | " |
| Carbon monoxide | 15,964 | " | " |
| Carbon dioxide | 4,169 | " | " |

The first three have the same heat capacity per cubic foot, so assuming their temperature 1550°:

| | | |
|---|---|---|
| $N^2$, $H^2$, CO | $76,011 \times 534.5 = 40,627,900$ | oz. Cal. |
| $CO^2$ | $4,169 \times 947.1 = 3,948,250$ | " |

$$44,576,150 \quad ''$$
$$= 2,786,000 \text{ lb. Cal.}$$

*Absorbed in Decomposing Moisture:* The 1,115 cubic feet of hydrogen in the gases represent so much steam or water vapor decomposed. Since 1 cubic foot = 0.09 ounces, the heat absorbed is

$$1,115 \times 0.09 \times 29,040 = 2,914,160 \text{ oz. Cal.}$$
$$= 182,130 \text{ lb. Cal.}$$

*Heat Conducted to the Air:* Assuming the conditions worked out in the illustration under this heading, this item would be approximately, for 9 min., 10 secs. in lb. Cal.

$$1,714 \times 2.204 \times 9.167 = 34,630 \text{ lb. Cal.}$$

*Heat Lost by Radiation:* Making similar assumption, we have

$$1,249 \times 2.204 \times 9.167 = 25,240 \text{ lb. Cal.}$$

| **Recapitulation.** | **Lb. Cal.** |
|---|---|
| Heat in converter body at starting | 8,034,970 |
|   "    melted pig-iron | 6,750,500 |
|   "    spiegeleisen | 750,000 |
|   "    blast | 86,580 |
| Heat of oxidation | 4,510,800 |
|   "    formation of slag | 59,890 |
| Total on hand and developed | 20,192,740 |
| Heat in converter body at finish | 7,183,770 |
|   "    finished steel | 8,632,750 |
|   "    slag | 1,582,350 |
|   "    escaping gases | 2,786,000 |
| Heat absorbed in decomposing moisture | 182,130 |
| Heat conducted to the air | 34,630 |
| Heat lost by radiation | 25,240 |
| Total accounted for | 20,426,870 |

Another way of expressing this balance is to itemize the avenues of heat evolution and utilization, as follows:

| | Lb. Cal. |
|---|---|
| Received from converter body | 851,200 |
| Received by oxidation | 4,510,800 |
| Received by formation of slag | 59,890 |
| Total | 5,421,890 |

| | Lb. Cal. |
|---|---|
| Used, excess of heat in gases over blast | 2,699,420 |
| Used, excess of heat in steel and slag over pig iron and spiegel | 2,714,600 |
| Decomposition of moisture | 182,130 |
| Radiation and conduction | 59,870 |
| Total | 5,556,020 |

[The next instalment of these calculations will discuss the relative heating efficiencies of different ingredients of the pig-iron, in the Bessemer process.]

## Tests of Titaniferous Slags.

By Chas. N. Cox, Jr., and Loring C. Lennox.

There are found in several places in the United States large bodies of iron ore of high grade, but containing titanic acid in considerable amounts. This ore is not being mined at present because of the prevailing prejudice among blast-furnace men against ores containing titanic acid, which is supposed to be the source of several difficulties in the working of the blast furnace, such as high smelting point and viscosity of the slag and the formation of accretions in the crucible (hearth).

Mr. A. J. Rossi, of New York, has constructed a small blast furnace and conducted quite an extensive series of experiments on the feasibility and fluidity of actual slag mixtures containing varying percents of titanic acid. His general conclusions were that it was perfectly feasible to obtain mixtures in the blast furnace that would produce slags fully as fusible and fluid and in every way as satisfactory as those obtained when using ore free from titanic acid. He attributed the trouble that this substance caused, in the early days of the iron manufacture in this country, to the ignorance of the blast-furnace men in regard to the material they were handling rather than the influence of the titanium. He thinks that the modern blast-furnace man's fear of titanium-bearing ores is based almost entirely upon prejudice, because of the early mistakes in the methods of handling them.

In Vol. III. of the Transactions of the American Institute of Mining Engineers, there is an article by Mr. Rossi, in which he states that in making up slags for a titanium-bearing ore, the titanic acid ($TiO_2$) should enter in as so much silica ($SiO_2$), and all calculations should be made with this in view.

In this article he gives the percentage composition of several slags which proved to be quite satisfactory, and which will appear later in this article as comparative tests. With these facts in view it was decided to make a number of tests on titaniferous slag mixtures in order to verify Mr. Rossi's experiments, and thus perhaps interest those who are in a position to do some experimental work on a commercial scale. Our tests were made in the Engineering Department of Colorado College.

Before attempting to outline the method of conducting these tests, it will be desirable to say something about the materials used in making up the slag. Of necessity the tests were conducted on a small scale, and so with the exception of the limestone and rutile (which formed the source of titanic acid), the ingredients were "C P" chemicals. Fortunately, a limestone was obtained, quarried in the Garden of Gods, which was, with the exception of 0.018 per cent $Fe_2O_3 + Al_2O_3$ and 0.0018 per cent MgO, pure $CaCO_3$. For the sake of simplifying the computations it was called 99.8 per cent pure, and the $Fe_2O_3 + Al_2O_3$ and MgO ignored.

The rutile, which was obtained from the American Rutile Co., of Washington, D. C., was found to contain:

Titanic acid $TiO_2$.................. 91.32%
Silica $SiO_2$.......................... 4.70
Ferrous oxide FeO................ 4.02
                                        _____
                                        100.4

As there was no guide to follow in the matter of types of mixtures to use, it was decided to test and compare the monosilicates, bisilicates, trisilicates and sesquisilicates. As all metallurgists know, the meanings of these prefixes are simply that the ratio of the oxygen atoms in combination with the bases of the slags to the oxygen atoms in combination with the acid-forming elements are respectively 1 to 1, 1 to 2, 1 to 3 and 2 to 3, and they are represented by the general formulæ $2RO . SiO_2$, $RO . SiO_2$, $2RO . 3SiO_2$ and $4RO . 3SiO_2$ respectively, when R represents any divalent base or bases. Also, $2R_2O_3 . 3SiO_2$, $R_2O_3 . 3SiO_2$, $2R_2O_3 . 9SiO_2$ and $4R_2O_3 . 9SiO_2$ are the corresponding formulæ when R represents trivalent bases. In the calculations that follow it has been assumed that $TiO_2$ may take the place of $SiO_2$ in these formulæ, and also that the bases in the slag divide themselves between these two acids in proportion to the respective masses of the acids. It was decided to use slag containing the following percentages of titanic acid: ½, 1, 3, 5, 7, 10, 14. In these slags alumina and calcium oxide in the ratio of 1 to 3 was used. In the slags containing ½ per cent and 1 per cent of $TiO_2$ the c.p. titanic acid was used; in the others rutile, the analysis of which was given above.

Two other sets of slags were tested, viz.: slags containing alumina, calcium oxide and magnesia, in the ratios 12 to 33 to 3, and the folowing per cents of titanic acid, 3, 7 and 14. Also slags containing alumina and calcium oxide in the ratio 1 to 4, and the following percents of titanic acid: 1, 3, 5, 7, 10, 14. The reasons for using these different ratios between the bases in the slags is that thus are obtained the percents of the bases usually found in ordinary blast furnace slags.

In the following tables will be found the results of computations for sixty-four different slags, and with the exception of the two columns following the one marked "total percent," they require no explanation. The column marked "percent rutile" was introduced for convenience in computing the weight of $TiO_2$ and FeO, required for a charge of any size of the slag. The column marked "percent raw," is inserted for similar reasons, the figures in this column being those representing the total percents of silica required less that contained in the rutile. All slags with 0.5 per cent and 1 per cent of $TiO_2$ were made up from c.p. $TiO_2$ without rutile.

To these tables is added one of four slag mixtures that are said by Mr. Rossi to behave in a satisfactory manner in the blast furnace; also the composition of a regular blast fur-

nace slag from one of the Jones & Loughlin Steel Co's blast furnaces, making Bessemer pig iron, and two of Prof. H. O. Hofman's ferrous silicates. These were used as giving a standard of comparison with actual practice and as checks on the accuracy of temperature readings.

The testing of these slags was conducted after two different methods, and with two distinct objects in view, viz.: to find the fusion points (temperatures) of the mixtures, and to find the comparative fluidity of the slags at a temperature of 1,410° C. For the fusion point test cones were used, or more properly speaking, triangular pyramids, as near the size of Prof. Orton's Seger cones as they could conveniently be made. This method of making the fusion tests was adopted because it seemed to be the one finding most ready acceptance at the present time for such work.

The first method tried for making the cones was mixing the



Front of Muffle
PLAN OF CONE ARRANGEMENT.

materials (ground to pass a 100-mesh screen) with a sufficient quantity of a syrup (made by boiling dextrine in water) to make a stiff paste, and then pressing this paste into a mould with a rammer made for the purpose. This method was unsatisfactory, because of the sticking of the material in the mould, and resort was finally made, after considerable experimenting, to rolling the mixture out into thin rolls and cutting out the cones with a spatula. It was also found, after several trials, that gum tragacanth was a much more preferable binding material; when mixed with the slag-forming materials it made a sort of dough, which was much easier to handle than the brash paste formed with dextrine. Then it was found that when the gum tragacanth cones were thoroughly dry, they could be placed immediately in a very hot furnace without breaking, while dextrine cones had to be heated up gradually, and the organic matter given time to burn off.

To test these cones they were placed erect in a dry fire-clay bed which was introduced into the muffle. The cones were arranged in rows, with a row of Prof. Orton's cones across the middle of the bed, as shown in the adjoining diagram.

The above figure shows the arrangement of the slag and standard cones in the furnace. Row No. 2 was usually the hottest part of the muffle, and the row of standard (Orton) cones occupied a position half-way between row No. 2 and row No. 5, which is the coolest row. In making up the bed for a second test the slag cones in rows 4 and 5 were interchanged with those in rows 1 and 2, so that by averaging the results obtained for any slag no great error, due to location in the furnace, will enter in. In this manner eight to twenty cones were tested of each slag. Six cones of each slag were made and tested, two at a time, so as to eliminate as far as possible any variations in temperature in different parts of the furnace.

The fluidity tests were made first in nickel crucibles, but it was found impossible to keep the atmosphere sufficiently reducing to prevent their burning through. Platinum ware was too expensive, so it was decided to use the ordinary fire-clay crucible, after first lining it with a mixture that was to be fused in it, and then glazing it by a preliminary heating. This method gave excellent results. In making these tests four of the glazed crucibles were placed with their charges of 20 grams each in the furnace, and a No. 14 Orton-Seger cone placed in

**TABLE No. 1.**
Ratio of lime to aluminum = 3 to 1.

| Silicate Type. | TiO₂ % | FeO % | SiO₃ % | Al₂O₃ % | CaO % | Total % | Rutile % | Raw SiO₃ % | Slag No. |
|---|---|---|---|---|---|---|---|---|---|
| Mono............... | .5 | .0 | 37.594 | 15.476 | 46.429 | 99.999 | .0 | 37.594 | 1 |
| Mono............... | 1. | .0 | 37.17 | 15.46 | 46.37 | 100. | .0 | 37.17 | 2 |
| Mono............... | 3. | .132 | 35.456 | 15.353 | 46.059 | 100. | 3.28 | 35.302 | 3 |
| Mono............... | 5. | .23 | 33.748 | 15.255 | 45.767 | 100. | 5.475 | 33.491 | 4 |
| Mono............... | 7. | .307 | 32.040 | 15.163 | 45.49 | 100. | 7.665 | 31.68 | 5 |
| Mono............... | 10. | .44 | 29.507 | 15.013 | 45.040 | 100. | 10.950 | 28.992 | 6 |
| Mono............... | 14. | .644 | 26.058 | 14.825 | 44.474 | 100.01 | 15.331 | 25.337 | 7 |
| Bi................. | .5 | .0 | 54.455 | 11.258 | 33.785 | 99.998 | .0 | 54.455 | 8 |
| Bi................. | 1. | .0 | 54.010 | 11.242 | 33.747 | 99.999 | .0 | 54.010 | 9 |
| Bi................. | 3. | .132 | 52.207 | 11.16 | 33.501 | 100. | 3.28 | 52.053 | 10 |
| Bi................. | 5. | .230 | 50.413 | 11.085 | 33.278 | 100. | 5.475 | 50.156 | 11 |
| Bi.......,....... | 7. | .307 | 48.617 | 11.014 | 33.062 | 100. | 7.665 | 48.257 | 12 |
| Bi................. | 10. | .44 | 45.929 | 10.903 | 32.728 | 100. | 10.95 | 45.414 | 13 |
| Bi................. | 14. | .644 | 42.329 | 10.753 | 32.274 | 100. | 15.331 | 41.608 | 14 |
| Tri................ | .5 | .0 | 64.087 | 8.853 | 26.56 | 100. | .0 | 64.087 | 15 |
| Tri................ | 1. | .0 | 63.628 | 8.842 | 26.528 | 99.998 | .0 | 63.628 | 16 |
| Tri................ | 3. | .132 | 61.776 | 8.773 | 26.319 | 100. | 3.28 | 61.622 | 17 |
| Tri................ | 5. | .230 | 59.814 | 8.739 | 26.219 | 100. | 5.475 | 59.557 | 18 |
| Tri................ | 7. | .307 | 58.086 | 8.652 | 25.955 | 100. | 7.665 | 57.726 | 19 |
| Tri................ | 10. | .44 | 56.085 | 8.369 | 25.105 | 100. | 10.95 | 55.57 | 20 |
| Tri................ | 14. | .644 | 53.299 | 8.014 | 24.043 | 100. | 15.331 | 52.578 | 21 |
| Sesqui............ | .5 | .0 | 47.372 | 13.032 | 39.096 | 100. | .0 | 47.372 | 22 |
| Sesqui............ | 1. | .0 | 46.936 | 13.016 | 39.049 | 100.01 | .0 | 46.936 | 23 |
| Sesqui............ | 3. | .132 | 45.17 | 12.952 | 38.773 | 100. | 3.28 | 45.016 | 24 |
| Sesqui............ | 5. | .230 | 43.410 | 12.839 | 38.521 | 100. | 5.475 | 43.153 | 25 |
| Sesqui............ | 7. | .307 | 41.652 | 12.760 | 38.281 | 100. | 7.665 | 41.292 | 26 |
| Sesqui............ | 10. | .44 | 39.014 | 12.636 | 37.91 | 100. | 10.95 | 38.499 | 27 |
| Sesqui............ | 14. | .644 | 35.491 | 12.466 | 37.399 | 100. | 15.331 | 34.770 | 28 |

**TABLE No. 2.**
Ratio of magnesia to lime to alumina = 3 to 12 to 33.

| Silicate Type. | TiO₂ % | FeO % | SiO₃ % | Al₂O₃ % | CaO % | MgO % | Rutile % | Raw SiO₃ % | Slag No. |
|---|---|---|---|---|---|---|---|---|---|
| Mono............... | 3. | .132 | 35.939 | 15.232 | 41.889 | 3.808 | 3.28 | 35.785 | 29 |
| Mono............... | 7. | .307 | 32.518 | 15.044 | 41.370 | 3.761 | 7.665 | 32.158 | 30 |
| Mono............... | 14. | .644 | 26.524 | 14.708 | 40.447 | 3.677 | 15.331 | 25.803 | 31 |
| Bi................. | 3. | .132 | 52.708 | 11.039 | 30.359 | 2.76 | 3.28 | 52.554 | 32 |
| Bi................. | 7. | .307 | 49.111 | 10.895 | 29.861 | 2.726 | 7.665 | 48.751 | 33 |
| Bi................. | 14. | .644 | 42.811 | 10.636 | 29.248 | 2.659 | 15.331 | 42.090 | 34 |
| Tri................ | 3. | .132 | 62.21 | 8.664 | 23.827 | 2.166 | 3.28 | 62.056 | 35 |
| Tri................ | 7. | .307 | 58.516 | 8.544 | 23.496 | 2.136 | 7.665 | 58.156 | 36 |
| Tri................ | 14. | .644 | 52.043 | 8.328 | 22.902 | 2.082 | 15.331 | 51.322 | 37 |
| Sesqui............ | 3. | .132 | 45.827 | 12.76 | 35.090 | 3.19 | 3.28 | 45.673 | 38 |
| Sesqui............ | 7. | .307 | 42.197 | 12.624 | 34.715 | 3.156 | 7.665 | 41.837 | 39 |
| Sesqui............ | 14. | .644 | 35.981 | 12.343 | 33.946 | 3.086 | 15.331 | 35.26 | 40 |

**TABLE No. 3.**
Ratio of lime to alumnia = 4 to 1.

| Type Silicate. | TiO₂ % | FeO % | SiO₃ % | Al₂O₃ % | CaO % | | | Slag No. |
|---|---|---|---|---|---|---|---|---|
| Mono............ | 1. | .0 | 36.575 | 12.485 | 49.94 | ........ | ........ | 41 |
| Mono............ | 3. | .132 | 34.864 | 12.402 | 49.602 | ........ | ........ | 42 |
| Mono............ | 5. | .230 | 33.16 | 12.322 | 49.288 | ........ | ........ | 43 |
| Mono............ | 7. | .307 | 31.457 | 12.248 | 48.988 | ........ | ........ | 44 |
| Mono............ | 10. | .44 | 28.93 | 12.126 | 48.504 | ........ | ........ | 45 |
| Mono............ | 14. | .644 | 25.489 | 11.974 | 47.893 | ........ | ........ | 46 |
| Bi............. | 1. | .0 | 53.41 | 9.117 | 36.473 | ........ | ........ | 47 |
| Bi............. | 3. | .132 | 51.612 | 9.051 | 36.205 | ........ | ........ | 48 |
| Bi............. | 5. | .230 | 49.821 | 8.989 | 35.959 | ........ | ........ | 49 |
| Bi............. | 7. | .307 | 47.834 | 8.972 | 35.887 | ........ | ........ | 50 |
| Bi............. | 10. | .44 | 43.347 | 8.842 | 35.371 | ........ | ........ | 51 |
| Bi............. | 14. | .644 | 41.757 | 8.72 | 34.879 | ........ | ........ | 52 |
| Tri............ | 1. | .0 | 63.063 | 7.187 | 28.75 | ........ | ........ | 53 |
| Tri............ | 3. | .132 | 61.213 | 7.13 | 28.525 | ........ | ........ | 54 |
| Tri............ | 5. | .230 | 59.371 | 7.079 | 28.319 | ........ | ........ | 55 |
| Tri............ | 7. | .307 | 57.533 | 7.032 | 28.128 | ........ | ........ | 56 |
| Tri............ | 10. | .44 | 54.693 | 6.973 | 27.894 | ........ | ........ | 57 |
| Tri............ | 14. | .644 | 51.087 | 6.854 | 27.415 | ........ | ........ | 58 |
| Sesqui.......... | 1. | .0 | 46.320 | 10.536 | 42.144 | ........ | ........ | 59 |
| Sesqui.......... | 3. | .132 | 44.559 | 10.462 | 41.847 | ........ | ........ | 60 |
| Sesqui.......... | 5. | .230 | 42.803 | 10.392 | 41.574 | ........ | ........ | 61 |
| Sesqui.......... | 7. | .307 | 41.049 | 10.329 | 41.315 | ........ | ........ | 62 |
| Sesqui.......... | 10. | .44 | 38.418 | 10.229 | 40.914 | ........ | ........ | 63 |
| Sesqui.......... | 14. | .644 | 34.904 | 10.091 | 40.361 | ........ | ........ | 64 |

Mr. Rossi's slags.

| No. | | | | | | Total. | | |
|---|---|---|---|---|---|---|---|---|
| 1 | 36.78 | 1.85 | 27.83 | 9.18 | 24.36 | ....... | ....... | .. |
| 2 | 25.11 | 3.46 | 26.72 | 11.86 | 25.81 | ....... | ....... | .. |
| 3 | 34.38 | 4.30 | 15.90-17.50 | 11.23 | 22.10 | ....... | ....... | .. |
| 4 | 64.80 | .90 | .67 | 10.50 | 14.30 | ....... | ....... | .. |

Slags investigated by Prof. H. O. Hofman.

| | | | | | | Melting Point Given by Prof. Hofman | |
|---|---|---|---|---|---|---|---|
| ............... | ....... | 40.81 | 19.19 | ....... | 40.00 | ....... | 1290° | .. |
| ............... | ....... | 31.55 | 24.45 | ....... | 44.00 | ....... | 1310° | .. |

Gray slag from blast furnace making Bessemer pig iron, Jones & Laughlin's Steel Company, Pittsburg, Pa.

| | | | | | | MgO | CaS | |
|---|---|---|---|---|---|---|---|---|
| ............... | ....... | .85 | 34.10 | 15.80 | 41.70 | 2.20 | 3.82 | .. |

their midst. The temperature was raised until the cone bent over, and a few minutes allowed for the mass in the crucible to become quiet, when it was removed from the furnace and immediately poured on a hot inclined slab. Tables VI. and VII. give the results of these tests.

The manner of noting the temperature of the fusion tests on the silicate cones has not been explained; it may be well to say that the cones were watched through a peep-hole in the door of the muffle, and the temperature of fusion of any particular slag determined by noticing which Orton-Seger cone was going down when it bent over. As will appear in the tables, the range of fusion points was within the limits of No. 7 cone (1,270° C.) to No. 11 (1,350° C.). It was also noticed that the slag cones bent much more rapidly than the Orton cones when they had started, so much so in fact, that if cones were put into a very hot furnace considerable difficulty was experienced in judging the fusion points.

In these experiments the standard cones used were made by Prof. Edward Orton, Jr., Columbus, Ohio. At this point it may be well to state that throughout the fusion tests the method set forth by Prof. H. O. Hofman, in his article entitled "Temperature of Formation of Various Silicates" (Trans. A. I. M. E., Vol. XXIX.) was followed.

At the outset of the work it was expected to make use of a Le Chatelier pyrometer, together with the Orton cones, but the only instrument at hand had the end of the porcelain tube broken off, so that the wires were exposed to the gases of the muffle. It was found that after the pyrometer had remained in the furnace for a little time, the readings would go down while the cone indicated that the furnace was as hot as before. Whatever may have been the cause of this drop in the pyrometer readings (it was ascribed to the absorption of gases by the wires) they were very unsatisfactory, even when attempts were made to correct them by cone tests, and so it was decided to discontinue their use.

Many difficulties were encountered in obtaining a temperature sufficient to do the work. First was tried a portable fireclay assay furnace, designed for use with anthracite or coke, but the highest temperature obtained with it was about 1,200° C., as against 1,500° C. desired. Attempts to secure an electrical resistance furnace failed, as no dealer could be found to furnish one that could be relied upon. A gasoline furnace failed to answer the requirements though shipped with a guarantee.

Finally, a small Sturtevant blower was connected by a 4-inch pipe to the ash pit of a stationary assay furnace with a deep fuel bed. With good coke packed around the muffle and forced draft from the blower, it was possible to attain a temperature of 1,500° in half an hour.

The high temperature used, especially in the fusibility tests, rapidly destroyed the muffle supports so that the muffles themselves were short lived, and much delay and many shut-downs were thus caused. Thus the work was at times discouraging,

and it was impossible in the limited time to perform as many experiments as were desirable.

The two sets of tables that follow need but little further explanation, but it may be well to say that a number of tests were made at first with the cones which were so unsatisfactory that the results were not entered in the tables. Instead of giving the temperature readings in terms of the Orton-Seger cones, they have been translated into degrees Cent.

In the column headed "remarks," in Table No. VI., will be noticed at intervals the expression "in the front of the muffle." This is not intended to mean in the very front part of the muffle (for the muffle was never filled to its capacity, and the charges were always placed in the back half), but rather those cones nearest the front, and in the front of the row of standard cones.

The muffle was small (11 inches x 7 inches x 5 inches), and it was found that at times when the coke did not work down along its sides evenly, or when the blast became partially choked at one point, there would often be as much as 50° C. difference in a distance of 6 inches. This cooling occurred almost always in the front part of the muffle, because at this point more difficulty was encountered in keeping the coke packed, due to the fact that the muffles were sunk into the brick work of the furnace about 8 inches, which made it difficult to use a poker effectively. Then, too, the front part of the furnace was more troublesome to keep free from accretions, and so the front part of the muffle was almost always cooler than the back part.

These difficulties have made the results much less exact and less uniform than is to be desired from a scientific point of view, but it is believed that remembering the plan of arranging the cones in successive tests, the average of the separate cones of each mixture is close enough for commercial purposes.

CaCO₃, in place of CaO, was used in making up the cones, this being done to avoid the slaking effects of the CaO, since the cones had to be made up wet; and it was thought that upon drying they would absorb CO₂ from the air of CaO were used, causing expansion and cracking. In the crucible tests, however, CaO was used, as these were to be made with dry mixtures. These latter were conducted at a temperature of 1,410° (No. 14 Orton cone), and the fused mass poured onto an iron plate, heated to about 300° C., and inclined at an angle of 7° 8″, the flow being measured along the plate.

1,410° C. was taken as the standard temperature for the fluidity tests, because it was the highest temperature that could be maintained without serious detriment to the furnace. In Table VI. the temperature readings marked * were taken by means of the Le Chatelier; readings not so marked were taken by means of the Orton cones.

### CONCLUSIONS.

From a study of the fusion tests it is noted that those cones containing magnesia fused at a temperature slightly less, gen-

## TABLE VI.

| Slag No. | Cone No. | Temp. | Remarks. |
|---|---|---|---|
| 1 | 1 | 1295°° | |
| 1 | 2 | 1295°° | |
| 1 | 3 | 1290°° | |
| 1 | 4 | 1290°° | |
| 1 | 5 | 1290°° | |
| 1 | 6 | 1290°° | |
| 2 | 1 | 1300°° | |
| 2 | 2 | 1300° | |
| 2 | 3 | 1305°° | |
| 2 | 4 | 1305°° | |
| 2 | 5 | 1300°° | |
| 2 | 6 | 1300°° | |
| 3 | 1 | 1290°° | |
| 3 | 2 | 1295°° | |
| 3 | 3 | 1260°° | |
| 3 | 4 | 1260°° | |
| 3 | 5 | 1300°° | |
| 3 | 6 | 1300°° | |
| 4 | 1 | 1295° | |
| 4 | 2 | 1300 | |
| 4 | 3 | 1280° | |
| 4 | 4 | | |
| 4 | 5 | 1290° | |
| 4 | 6 | 1290° | |
| 5 | 1 | 1292°° | |
| 5 | 2 | 1292°° | |
| 5 | 3 | 1280°° | |
| 5 | 4 | 1280°° | |
| 5 | 5 | 1305°° | |
| 5 | 6 | 1320°° | |
| 6 | 1 | 1300°° | |
| 6 | 2 | 1270°° | |
| 6 | 3 | 1270°° | |
| 6 | 4 | 1270°° | |
| 6 | 5 | 1270°° | |
| 6 | 6 | 1270°° | |
| 7 | 1 | 1320°° | |
| 7 | 2 | 1320°° | |
| 7 | 3 | 1270°° | |
| 7 | 4 | 1270°° | |
| 7 | 5 | 1330°° | |
| 7 | 6 | 1330°° | |
| 8 | 1 | 1310°° | |
| 8 | 2 | 1330°° | |
| 8 | 3 | 1320°° | |
| 8 | 4 | | Broken. |
| 8 | 5 | | Broken. |
| 8 | 6 | | Broken. |
| 9 | 1 | 1305°° | |
| 9 | 2 | 1315°° | |
| 9 | 3 | 1295°° | |
| 9 | 4 | 1300°° | |
| 9 | 5 | | Broken. |
| 9 | 6 | | Broken. |
| 10 | 1 | 1270°° | |
| 10 | 2 | 1270°° | |
| 10 | 3 | 1330° | |
| 10 | 4 | 1330° | |
| 10 | 5 | | Broken. |
| 10 | 6 | | Broken. |
| 11 | 1 | 1330° | |
| 11 | 2 | 1330° | |
| 11 | 3 | 1270° | |
| 11 | 4 | 1270° | |
| 11 | 5 | 1270° | |
| 11 | 6 | 1270° | |
| 12 | 1 | 1310° | |
| 12 | 2 | 1310° | |
| 12 | 3 | 1310° | |
| 12 | 4 | 1310° | |
| 12 | 5 | 1290° | |
| 12 | 6 | 1290° | |
| 13 | 1 | 1270° | |
| 13 | 2 | 1310° | |
| 13 | 3 | 1270° | |
| 13 | 4 | 1270° | |
| 13 | 5 | 1270° | |
| 13 | 6 | 1270° | |
| 14 | 1 | 1290° | |
| 14 | 2 | 1270° | |
| 14 | 3 | 1270° | |
| 14 | 4 | 1270° | |
| 14 | 5 | 1310° | |
| 14 | 6 | 1290° | |
| 15 | 1 | 1332°° | |
| 15 | 2 | 1332°° | |
| 15 | 3 | 1350°° | |
| 15 | 4 | 1350°° | |
| 15 | 5 | | Broken. |
| 15 | 6 | | Broken. |
| 16 | 1 | 1290° | |
| 16 | 2 | 1290° | |
| 16 | 3 | 1290° | |
| 16 | 4 | 1290° | |
| 16 | 5 | 1290° | |
| 16 | 6 | 1290° | |
| 17 | 1 | 1310° | |
| 17 | 2 | 1310° | |
| 17 | 3 | 1310° | |
| 17 | 4 | 1290° | |
| 17 | 5 | 1270° | Cones put in hot muffle standard cones did not respond readily enough to give satisfactory reading. |
| 17 | 6 | 1270° | |
| 18 | 1 | 1310° | |
| 18 | 2 | 1310° | |
| 18 | 3 | 1270° | |
| 18 | 4 | 1270° | |
| 18 | 5 | 1290° | |
| 18 | 6 | 1290° | |
| 19 | 1 | 1270° | |
| 19 | 2 | 1270° | |
| 19 | 3 | 1290° | |
| 19 | 4 | 1290° | |
| 19 | 5 | 1290° | |
| 19 | 6 | 1270° | |
| 20 | 1 | 1310° | |
| 20 | 2 | 1310° | |
| 20 | 3 | 1270° | Furnace too hot when cones put in. Reading not satisfactory. |
| 20 | 4 | 1270° | |
| 20 | 5 | 1270° | |
| 20 | 6 | 1270° | |
| 21 | 1 | 1290° | |
| 21 | 2 | 1290° | |
| 21 | 3 | 1270° | |
| 21 | 4 | 1270° | |
| 21 | 5 | 1270° | Same as above. |
| 21 | 6 | 1270° | |
| 22 | 1 | 1290° | |
| 22 | 2 | 1290° | |
| 22 | 3 | 1330° | In front of muffle. |
| 22 | 4 | 1350° | |
| 22 | 5 | 1270° | |
| 22 | 6 | 1270° | |
| 23 | 1 | 1310° | |
| 23 | 2 | 1310° | |
| 23 | 3 | 1270° | |
| 23 | 4 | 1270° | |
| 23 | 5 | 1270° | |
| 23 | 6 | 1270° | |
| 24 | 1 | 1310° | |
| 24 | 2 | 1290° | |
| 24 | 3 | 1290° | |
| 24 | 4 | 1290° | |
| 24 | 5 | 1270° | Readings unsatisfactory, muffle too hot when cones put in. In front row. |
| 24 | 6 | 1270° | |
| 25 | 1 | 1290° | |
| 25 | 2 | 1290° | |
| 25 | 3 | 1290° | |
| 25 | 4 | 1290° | |
| 25 | 5 | 1290° | |
| 25 | 6 | 1290° | |
| 26 | 1 | 1290° | |
| 26 | 2 | 1290° | |
| 26 | 3 | 1290° | |
| 26 | 4 | 1290° | |
| 26 | 5 | 1290° | |
| 26 | 6 | 1290° | |
| 27 | 1 | 1290° | |
| 27 | 2 | 1290° | |
| 27 | 3 | 1290° | |
| 27 | 4 | 1290° | |
| 27 | 5 | 1290° | |
| 27 | 6 | 1290° | |
| 28 | 1 | 1290° | |
| 28 | 2 | 1290° | |
| 28 | 3 | 1290° | |
| 28 | 4 | 1290° | |
| 28 | 5 | 1270° | |
| 28 | 6 | 1290° | |
| 29 | 1 | 1310° | In front of muffle. |
| 29 | 2 | 1310° | |
| 29 | 3 | 1290° | |
| 29 | 4 | 1290° | |
| 29 | 5 | 1290° | |
| 29 | 6 | 1290° | |
| 30 | 1 | 1310° | In front of muffle. |
| 30 | 2 | 1310° | |
| 30 | 3 | 1290° | |
| 30 | 4 | 1290° | |
| 30 | 5 | 1290° | |
| 30 | 6 | 1290° | |
| 31 | 1 | 1290° | |
| 31 | 2 | 1290° | |
| 31 | 3 | 1290° | |
| 31 | 4 | 1290° | |
| 31 | 5 | 1270° | |
| 31 | 6 | 1270° | |
| 32 | 1 | 1310° | |
| 32 | 2 | 1310° | |
| 32 | 3 | 1290° | |
| 32 | 4 | 1270° | |
| 32 | 5 | 1270° | |
| 32 | 6 | 1270° | |
| 33 | 1 | 1310° | |
| 33 | 2 | 1310° | |
| 33 | 3 | 1290° | |
| 33 | 4 | 1290° | |
| 33 | 5 | 1290° | |
| 33 | 6 | 1290° | |
| 34 | 1 | | |
| 34 | 2 | 1290° | |
| 34 | 3 | 1290° | |
| 34 | 4 | 1290° | |
| 34 | 5 | 1290° | |
| 34 | 6 | 1290° | |
| 35 | 1 | 1310° | In front of muffle |
| 35 | 2 | 1310° | |
| 35 | 3 | 1310° | |
| 35 | 4 | 1310° | |
| 35 | 5 | 1290° | |
| 35 | 6 | 1290° | |
| 36 | 1 | 1290° | In front of muffle. |
| 36 | 2 | 1310° | |
| 36 | 3 | 1290° | |
| 36 | 4 | 1290° | |
| 36 | 5 | 1290° | |
| 36 | 6 | 1290° | |
| 37 | 1 | 1290° | |
| 37 | 2 | 1290° | |
| 37 | 3 | 1270° | |
| 37 | 4 | 1270° | |
| 37 | 5 | 1290° | |
| 37 | 6 | 1290° | |
| 38 | 1 | 1290° | In front of muffle. |
| 38 | 2 | 1290° | |
| 38 | 3 | 1270° | |
| 38 | 4 | 1270° | |
| 38 | 5 | 1270° | |
| 38 | 6 | 1270° | |
| 39 | 1 | 1290° | In front of muffle. |
| 39 | 2 | 1290° | |
| 39 | 3 | 1270° | |
| 39 | 4 | 1290° | |
| 39 | 5 | 1270° | |
| 39 | 6 | 1270° | |
| 40 | 1 | 1290° | |
| 40 | 2 | 1290° | In front rows. |
| 40 | 3 | 1270° | |
| 40 | 4 | 1290° | |
| 40 | 5 | 1270° | |
| 40 | 6 | 1270° | |
| 41 | 1 | 1310° | |
| 41 | 2 | 1310° | |
| 41 | 3 | 1290° | |
| 41 | 4 | 1290° | |
| 41 | 5 | 1290° | |
| 41 | 6 | 1290° | |
| 42 | 1 | 1310° | Reading unsatisfactory, bending of cone not noticed at first. When observed, seyer No. 9 was going down. |
| 42 | 2 | 1310° | |
| 42 | 3 | 1290° | |
| 42 | 4 | 1290° | |
| 42 | 5 | 1270° | |
| 42 | 6 | 1290° | |
| 43 | 1 | 1310° | Reading unsatisfactory. Same as above. |
| 43 | 2 | 1310° | |
| 43 | 3 | 1290° | |
| 43 | 4 | 1290° | |
| 43 | 5 | 1291° | |
| 43 | 6 | 1310° | |
| 44 | 1 | 1310° | Same as 42, 1 and 2. |
| 44 | 2 | 1310° | |
| 44 | 3 | 1290° | |
| 44 | 4 | 1290° | |
| 44 | 5 | 1290° | |
| 44 | 6 | 1290° | |
| 45 | 1 | 1310° | |
| 45 | 2 | 1310° | |
| 45 | 3 | 1310° | |
| 45 | 4 | 1310° | |
| 45 | 5 | 1310° | |
| 45 | 6 | 1310° | |
| 46 | 1 | 1310° | |
| 46 | 2 | 1310° | |
| 46 | 3 | 1290° | |
| 46 | 4 | 1290° | |
| 46 | 5 | 1310° | |
| 46 | 6 | 1310° | |
| 47 | 1 | 1310° | |
| 47 | 2 | 1310° | |
| 47 | 3 | 1350° | In front rows. |
| 47 | 4 | 1350° | |
| 47 | 5 | 1310° | |
| 47 | 6 | 1310° | |
| 48 | 1 | 1330° | In front rows. |
| 48 | 2 | 1330° | |
| 48 | 3 | 1290° | |
| 48 | 4 | 1290° | |
| 48 | 5 | 1290° | |
| 48 | 6 | 1290° | |
| 49 | 1 | 1330° | In front rows. |
| 49 | 2 | 1330° | |
| 49 | 3 | 1290° | |
| 49 | 4 | 1290° | |
| 49 | 5 | 1290° | |
| 49 | 6 | 1290° | |
| 50 | 1 | 1330° | In front rows. |
| 50 | 2 | 1330° | |
| 50 | 3 | 1290° | |
| 50 | 4 | 1290° | |
| 50 | 5 | 1290° | |
| 50 | 6 | 1290° | |
| 51 | 1 | 1290° | |
| 51 | 2 | 1290° | |
| 51 | 3 | 1290° | |
| 51 | 4 | 1290° | |
| 51 | 5 | 1310° | |
| 51 | 6 | 1310° | |
| 52 | 1 | 1290° | |
| 52 | 2 | 1290° | |
| 52 | 3 | 1290° | |
| 52 | 4 | 1290° | |
| 52 | 5 | 1310° | |
| 52 | 6 | 1310° | |
| 53 | 1 | 1350° | |
| 53 | 2 | 1350° | Only slightly fused. |
| 53 | 3 | 1350° | |
| 53 | 4 | 1350° | |
| 53 | 5 | 1310° | |
| 53 | 6 | 1310° | |
| 54 | 1 | 1310° | |
| 54 | 2 | 1310° | |
| 54 | 3 | 1290° | |
| 54 | 4 | 1290° | |
| 54 | 5 | 1310° | |
| 54 | 6 | 1310° | |
| 55 | 1 | 1310° | |
| 55 | 2 | 1310° | |
| 55 | 3 | 1290° | |
| 55 | 4 | 1290° | |
| 55 | 5 | 1290° | |
| 55 | 6 | 1290° | |
| 56 | 1 | 1290° | |
| 56 | 2 | 1290° | |
| 56 | 3 | 1290° | |
| 56 | 4 | 1290° | |
| 56 | 5 | 1290° | |
| 56 | 6 | 1290° | |
| 57 | 1 | 1310° | |
| 57 | 2 | 1310° | |
| 57 | 3 | 1290° | |
| 57 | 4 | 1290° | |
| 57 | 5 | 1290° | |
| 57 | 6 | 1290° | |
| 58 | 1 | 1330° | |
| 58 | 2 | 1330° | |
| 58 | 3 | 1290° | |
| 58 | 4 | 1290° | |
| 58 | 5 | 1290° | |
| 58 | 6 | 1290° | |
| 59 | 1 | 1330° | |
| 59 | 2 | 1330° | |
| 59 | 3 | 1290° | |
| 59 | 4 | 1290° | |
| 59 | 5 | 1310° | |
| 59 | 6 | 1310° | |
| 60 | 1 | 1330° | |
| 60 | 2 | 1330° | |
| 60 | 3 | 1310° | |
| 60 | 4 | 1310° | |
| 60 | 5 | 1310° | |
| 60 | 6 | 1310° | |
| 61 | 1 | 1290° | |
| 61 | 2 | 1290° | |
| 61 | 3 | 1310° | |
| 61 | 4 | 1290° | |
| 61 | 5 | 1290° | |
| 61 | 6 | 1290° | |
| 62 | 1 | 1310° | |
| 62 | 2 | 1310° | |
| 62 | 3 | 1290° | |
| 62 | 4 | 1290° | |
| 62 | 5 | 1310° | |
| 62 | 6 | 1310° | |
| 63 | 1 | 1310° | |
| 63 | 2 | 1310° | |
| 63 | 3 | 1290° | |
| 63 | 4 | 1290° | |
| 63 | 5 | 1310° | |
| 63 | 6 | 1310° | |
| 64 | 1 | 1310° | |
| 64 | 2 | 1310° | |
| 64 | 3 | 1290° | |
| 64 | 4 | 1290° | |
| 64 | 5 | 1310° | |
| 64 | 6 | 1310° | |

Gray blast furnace slag. Jones & Laughlin's Fur. making Bessemer iron. } 1290° / 1290°

Silicates given in Prof. Hofmans's article.

| Slag No. | Cone No. | Temp. | Remarks. |
|---|---|---|---|
| 1 | 1 | 1290° | These temperatures agree exactly with ones given by Prof. Hofman. |
| 1 | 1 | 1290° | |
| 2 | 1 | 1310° | |
| 2 | 2 | 1310° | |

## TABLE VII.
### FLUIDITY TESTS.

| Slag No. | |
|---|---|
| 1 | Glassy white and very viscous. |
| 2 | Glassy white and very viscous. |
| 3 | Glassy, brown and mottled; flowed 1½″. |
| 4 | Mottled brown glass; flowed 1½″. |
| 5 | Mottled brown glass; flowed 1 1-16″. |
| 6 | Dark brown, glassy; flowed 2½″. |
| 7 | Dark brown, glassy; flowed 2½″. |
| 8 | Glassy, greenish, very slightly fluid. |
| 9 | Glassy, greenish, very slightly fluid. |
| 10 | Stony mass. Viscous; would not flow. |
| 11 | Glassy, viscous — Tests made in nickel |
| 12 | Glassy, viscous — crucible and poured |
| 13 | Glassy, viscous — immediately on |
| 14 | Glassy, viscous — reaching 1410°. Not satisfactory tests. |
| 15 | White, dead surface, no flow. |
| 16 | White, dead surface, no flow. |
| 17 | Glassy, viscous. |
| 18 | Glassy, viscous. |
| 19 | Glassy, light brown, very viscous. |
| 20 | Glassy, gray, very viscous. |
| 21 | Glassy, brown, viscous. |
| 22 | Stony, gray, no flow. |
| 23 | Glassy, green, viscous. |
| 24 | Glassy, brown, viscous. |
| 25 | Glassy, milky, viscous. |
| 26 | Glassy, dark brown; flowed 1½″. |
| 27 | Glassy, dark brown; flowed 2 1-16″. |
| 28 | Glassy, dark brown; flowed 1 15-16″. |
| 29 | Glassy, very light brown; flowed 1½″. |
| 30 | Glassy, mottled brown; flowed 1½″. |
| 31 | Glassy, mottled brown; flowed 1½″. |
| 32 | Glassy, gray, very viscous. |
| 33 | Glassy, dark opaque, viscous. |
| 34 | Glassy, mottled, black; flowed 1½″. |
| 35 | Gray, earthy mass; no flow. |
| 36 | Gray, earthy mass; no flow. |
| 37 | Stony; no flow. |
| 38 | Glassy, mottled green, viscous. |
| 39 | Glassy, opaque, mottled viscous. |

Slag
No.
40 Glassy, black; flowed 1¾".
41 Glassy, clear light green; flowed 1½".
42 Glassy, clear light green; flowed 1½".
43 Gray, opaque mottled glass; flowed 1½".
44 Gray, opaque mottled glass; flowed 1½".
45 Dark glass; flowed 2".
46 Black stony mass; flowed 2½".
47 Stony, gray mass; no flow | In front
48 Stony, gray mass; no flow | row of muffle.
49 Glassy, dark green, viscous.
50 Glassy, dark green, viscous.

Slag
No.
51 Glassy, brown, opaque; flowed 1".
52 Glassy, brown, opaque; flowed 1".

Rossi's Slags.
1 Gray, stony; flowed 1½".
2 Black glass; flowed 2½". Scoured crucible.
3 Dark gray earth mass; no flow.
4 Black stony mass; no flow.

Pulverized Gray Blast Furnace Slag.
1 Would not pour out of crucible.
2 Would not pour out of crucible.

erally speaking, than the others; also that the cones containing lime and alumina in the ratio 4 to 1 fused at a temperature slightly higher than those containing the same elements in the ratio 3 to 1. These differences were, however, slight, and since the fusion points of all the slags come within the range 1,270° C. to 1,350° C. (this being considerably below the temperature existing in the bosh and hearth of the iron blast furnace), we consider ourselves justified in saying that no difficulty would be encountered in fusing any of these mixtures in the ordinary blast furnace, but their fluidity after fusion is a question still to be considered.

In these cone tests very little, if any real differences, were found in the fusion points of the different silicates tested. The character of the slag was noted in each case, but this varies so largely with the location of the cones in the muffle, the time that they remained in (waiting for others of the same tests to fuse), and the maximum temperature reached before they were removed, that it can give no real information in regard to the comparative merits of the slags.

An idea may be had of the accuracy with which the temperatures of fusion were taken by a glance at the work on Prof. Hofman's silicates, given in Table No. VI.

From a study of the results obtained by the fluidity tests it is seen that the monosilicates are the most fluid, the bisilicates and sesqisilicates are quite viscous, and the trisilicates containing the higher percentages of $TiO_2$ were somewhat fluid.

Little difference was found in slags made up with varying proportions of bases; but it was found that the fluidity almost invariably increased with the increase of $TiO_2$. It is to be noted, however, that owing to the use of rutile with the increase of $TiO_2$, there was also a slight increase of the percentage of FeO, though the total amount of FeO was not higher than is frequently found in blast-furnace slags.

In no case were the comparison slags found to be more fluid than our own; and since these comparison slags have proved to be all that is required in blast furnace work, we see no reason why the more fluid, at least, of our slags might not be used successfully in a blast furnace.

Of course, it was impossible to judge accurately from the small charges used, what would be the action of large masses of a slag. In our case the small quantity of intensely hot material was bound to chill very quickly upon coming into contact with the air; and then to add to this, the hot plate we used was heated by gas, and 300° C. was about as hot as it could be kept, so that it was cold to the slag.

Our main object in performing these experiments was, if possible, to prove that $TiO_2$ in an ore, if properly handled, is no great detriment. There is good authority for the statement that blast furnaces have been run advantageously on ores containing as high as 5 per cent $TiO_2$.

Now we have found that apparently the fluidity of our slags increased with the increase in the percent of titanic acid. Therefore, we believe that $TiO_2$ in moderate quantity in ore, if properly handled in a blast furnace, should give less trouble than has been heretofore supposed.

Colorado College,
Colorado Springs, Col.

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# SYNOPSIS OF PERIODICAL LITERATURE.

**A Summary of Articles Appearing in American and Foreign Periodicals.**

### CHEMICAL AND ELECTROCHEMICAL ENGINEERING.

**Platinized Platinum Anodes for Chlorate Production.**—In the electrolysis of sodium and potassium chlorides platinized platinum anodes are not yet used commercially, although Foerster found that not only the voltage at the same is lower by 0.5 or 0.6 volts than at polished platinum electrodes, but that the ampere-hour efficiency in the production of hypochlorites and chlorates is thereby considerably increased. The chief reason why platinized-platinum electrodes have not found favor in practice is that they were not durable. W. Geibel describes in ,ne *Zeit. f. Elecktrochemie*, Nov. 9, experiments made in the laboratory of the well-known platinum works of Heraeus. The chief result is that if the platinum-black layer on the electrode is transformed into platinum-gray by careful heating, the layer sticks firmly to the electrode and the latter is sufficiently durable for use in practice. At the same time the platinum-gray electrode gives about the same increase of efficiency over the polished plate as can be obtained with platinum-black. The transformation of platinum-black into platinum-gray is best carried out in an electric furnace, the temperature of which may be exactly controlled. Since it is important that the effective surface is as large as possible, the heating temperature should not be too high.

**Switchboard for Electrolytic Analysis.**—In the *Engineering and Mining Journal* of Nov. 17, E. Larison describes a switchboard attachment which he has found useful for the electrolytic determination of copper. The source of current is a three-cell storage battery charged periodically from a 125-volt direct-current lighting circuit. To reduce the voltage a bank of twenty lamps of 16 cp. in parallel is placed in series with the battery. Two separate circuits in parallel are provided on the switchboard, each including a number of resistances in series. To obtain a close control of the current supplied to the individual solutions undergoing electrolysis, a second auxiliary switchboard is used. The connections are shown in diagram.

**Electro-Analysis.**—A paper by J. R. Withrow on the electrolytic precipitation of gold, with the use of a rotating anode, is found in the October issue of the *Journal* of the American Chemical Society. The author experimented with solutions of auric chloride in the presence of, first, potassium cyanide, and, second, sodium sulphide. Pure potassium cyanide was found to be a better electrolyte than sodium sulphide for this particular purpose.

**Concentration of Sulphuric Acid.**—It is a well-known fact that the concentration of sulphuric acid in platinum vessels of large size requires a very costly plant, while the use of porcelain pots is attended with considerable troubles, due to the breakage of the pots. A concentration furnace of modern construction, designed by G. Krell, which depends upon the use of cast iron pipes, is described by P. Schuberg in the *Zeitschrift für Chemische Apparatenkunde* May 15. The apparatus takes acid which has been concentrated previously to 62° B. in lead vessels. The concentration of the acid takes place in two horizontal cast iron pipes, which are half filled with the acid, which passes through them successively. The pipes are made of a special brand of cast iron, dense and free from blow holes,

and are arranged in a cast iron pan, which is filled with molten lead; the surface of the latter must be at least 30 millimeters above the cast iron pipes. The joint between the pipes and the wall of the cast iron vessel is made in such a manner as to allow for expansion and contraction, while at the same time preventing the escape of molten lead. Each pipe is 2,500 mm. long, 200 mm. wide and 20 mm. thick, and an even thickness at all parts of the pipe is of prime importance, as it allows the turning of the pipe around its axis so as to expose always fresh surfaces to the action of the acid. The acid attacks the first pipe to a slight extent, especially at the level of the liquid, but the second pipe suffers hardly any attack. At the fire-box side of the furnace the two pipes are joined by a cast iron pipe, which near the first pipe is provided with a baffle plate dipping below the level of the acid. By this means the distillate from each pipe can be collected separately. The apparatus is also provided with a sediment collector, and means are arranged for scraping the sediment out of the concentrating pipes. A considerable quantity of sediment is formed in them, and it has to be removed regularly and conveniently while the furnace is running. At several places, notably at the feed pipe and the discharge pipe of the acid, lead has been used, which is exposed to quite an elevated temperature, although it is cooled. If this were ordinary lead it would decompose in 60° acid at temperatures between 204° and 232° C. The lead used in this apparatus is, however, alloyed with some metal, which makes it more resistant towards hot sulphuric acid and at the same time increases its melting point to 320° C. without decomposition in sulphuric acid. The plant at Bruchhausen-Hüsten, Germany, which occupies only a small space and concentrates glover and waste acid to 92/93 and 97/98 per cent acid, produces in 24 hours 7,500 kg. of 92/93 per cent, or 3,500-4,000 kg. of 97/98 per cent acid. The last named figure is reached when the acid entering into the apparatus has a strength of 60-62° B., and a temperature of 170-180° C. The cast iron pipes last for 300-500 tons of 92 per cent, or 200-300 tons of 98 per cent acid. The cost of running decreases with the increasing number of furnaces, as one man can attend to three furnaces. The cost during a considerable period of practical operation, per 100 kg. finished 98 per cent acid, was as follows: With one double bath, labor, two shifts at $1.00, 5 cents; 19 kg. coal (100 kg. at 37.5 cents), 7 cents; wear and tear and repairs, 1.75 cents; a total of 13.75 cents. With two double baths the cost is decreased to 11 cents, and with three double baths to 10.5 cents. For the 92/93 per cent acid the cost decreases in accordance with the greater productive capacity of the furnace to 9 to 10 cents per 100 kg.

**Deacon Chlorine Process.**—In the well-known Deacon chlorine process a mixture of air and hydrochloric acid gas is led over a rough surface impregnated with cupric chloride, kept at a temperature of about 400° C. The cupric chloride acts only as a catalyzer, and may be replaced by the chloride of iron, nickel or cerium. The reaction is simply $4HCl + O_2 = 2H_2O + 2Cl_2$. The reaction never runs completely to an end, but approaches a state of equilibrium which, according to the conditions of temperature and concentration, fixes a definite limit to the yield of chlorine which it is possible to attain. G. N. Lewis has studied, in a paper in the October issue of the *Journal* of the American Chemical Society, this equilibrium in the Deacon process. In his experiments the catalyzer cupric chloride required always several days to reach a constant state, after which concordant results were obtained. The time required to bring the gases to equilibrium was greater the lower the temperature, and the smaller the percentage of oxygen present. Experiments were made at three temperatures, 352°, 386° and 419° C., and over a considerable range of concentration, the ratio of oxygen to hydrochloric acid in the final mixture being forty times as great in one case as in another. At constant temperature the results are in accordance with the mass law. The yield of chlorine from a given mixture is

greater the lower the temperature, and the change of the equilibrium constant with the temperature is in good agreement with the equation of Van't Hoff. The results may be seen from the following table:

| Temp. Degrees C. | Average Pressure in Atmospheres. | Mols. $O_2$ to 100 Mols. HCl in Original Gas. | Mols. $O_2$ to 100 Mols. HCl in Final Gas. | x. | K. |
|---|---|---|---|---|---|
| 352 | 1.00 | 92.7 | 544.0 | 0.869 | 4.15 |
| 352 | 0.93 | 29.7 | 49.5 | 0.808 | 3.95 |
| 386 | 0.98 | 327.0 | 1970.0 | 0.845 | 2.94 |
| 386 | 0.96 | 48.8 | 146.0 | 0.804 | 3.01 |
| 419 | 1.08 | 327.0 | 1700.0 | 0.820 | 2.40 |

In this table $x$ represents the fraction of the hydrochloric acid decomposed, or, in other words, the ratio of free chlorine to total chlorine in the final mixture. K is the equilibrium constant of the mass law, according to the equation:

$$K = \left( P_{Cl_2}^{\frac{1}{2}} \quad P_{H_2O}^{\frac{1}{2}} \right) \div \left( P_{HCl} \quad P_{O_2}^{\frac{1}{4}} \right)$$

An application is finally made to a calculation of the e. m. f. of the hydrogen-oxygen cell, which is found to be over 1.2 volt at 25°. This confirms the author's former conclusion that the value at present accepted for this e. m. f. is 0.1 volt too low.

### METALLURGY.

**Removal of Wood in Ore Dressing.**—It is a well-known fact that the presence of wood in the ore occasionally gives rise to considerable trouble in the concentrating mills. Mr. A. H. Wethey, in the *Engineering and Mining Journal*, Oct. 20, discusses the method used at the Butte Reduction Works, Butte,
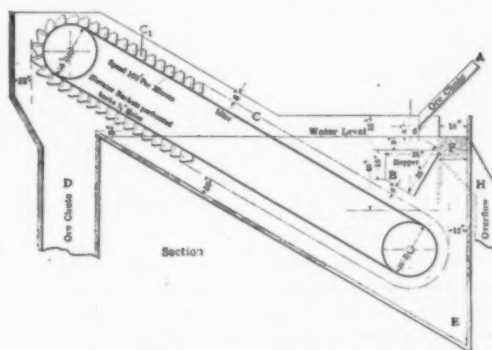


FIG. I.—SEPARATOR FOR REMOVING WOOD.

Mont., for getting rid of the wood. It has been found by actual experience at that plant that an ordinary mine wedge, 8 inches long, 4 inches wide, and 2 inches thick at the large end, when caught in a 9 x 15-inch crusher, will take as long to work its way through the crusher as a ton of ore. The ore at the above plant passes to a 15 x 24-inch crusher, set to crush down to a minimum of about 2½ inches. No trouble is caused by the presence of wood in this crusher. Previous to passing over this crusher the fine material has been screened off from the ore by means of a grizzley and any iron or steel, such as hammer heads, ends of drill steel, etc., have been removed by a magnet. The ore then passes through trommels, screening out the 1½-inch material, the over-size going to two 9 by 15-inch Blake crushers. The greatest difficulty was experienced with the pieces of wood in these crushers, and a novel wood-separating device was therefore installed. It is shown in Fig. 1. The ore is delivered into the device by the chute A, which passes it into a hopper B below the water level. From there the ore passes into the conveyor C, which in turn brings it to the ore chute D. As the tank E is kept full of water, any wood in the ore, when the latter is discharged into the hopper B, will float to the top of the water. The elevator buckets C on the con-

veyor have perforated backs, through which the water flows out as the elevator lifts the ore out of the water. The water, therefore, rushes back into the tank with sufficient force to wash the floating wood into the screen F, which removes the water, the wood falling down the launder G and the water pouring over the overflow H. It has been found by actual experience that from 20,000 tons of ore there are removed by means of this device 40 tons of wood, and the author estimates that by removing the wood the capacity of the crushing plant is increased 15 per cent.

### GOLD.

**Tube Mill Practice.**—The discussion of Mr. Darling's paper before the Chemical, Metallurgical and Mining Society of South Africa, mentioned previously in these abstracts, has touched several interesting questions connected with tube milling. In the September issue of the *Journal* of that Society, Mr. H. T. Brett explains the difference in the wear of the liners in Australia tube mills and those on the Rand. As far as the mills themselves are concerned, he remarks that the largest size mill in Western Australia is only 16 feet long, and only a few of that size are in use, while the majority are 13 feet long. This fact is due to the greater economy shown by the small tube mills. The in-take of these mills is bell shaped, the feed itself entering through a 2½-inch pipe tapered to 1½ inches at the end, so as to form a nozzle. This pipe forms the bottom discharge of a cone-shaped spitzkasten, usually 6 feet deep, the pressure forcing the feed through the nozzle into the mill. A round plate about 12 inches in diameter at the discharge end, and with long rectangular slots cut in it, was usually furnished with the mill, but it has been replaced by a ⅜-inch plate perforated with ⅛-inch holes. The liners are fastened to the shell with bolts. In the event of a bolt dropping through or working loose, a wooden plug is driven in, which keeps the liner tight until it is replaced. Eight hours' work of three men is required to take off an old set, replace it with new liners and to start the mill again. As far as the author is aware no silex liners have ever been tried. The chilled liners are 1 inch thick, cast in most cases at the mine; they last from four to six months. It is quite remarkable that these 1-inch chilled liners last longer than, say, a 4-inch silex or manganese steel liner, and the author traces the causes of this surprising fact. In the first place he points out that there is quite a difference in the coarseness of the sand which enters the mill. Nearly all the Kalgoorlie mines are grinding the total produce to a slime, and the fine sand is returned again and again to the tube mill until 95 per cent of it passes a 150-mesh screen. On the Rand, however, the mills are grinding coarse sand to fine in practically one passage through the mill, and therefore the feed entering a Kalgoorlie mill, being composed of, say, one part of original coarse sand from the battery to ten parts of returned fine sand, is in a much finer state of division than that entering the large mills on the Rand. The author cites as a further proof of the influence of the coarse sand in the life of the liners, the fact that 1 inch liners last six months in some of the mines, where the coarse sand is broken down in grinding pans first and then finished off in tube mills. With reference to the tonnage passed through the mill per day as a factor in the wear of the liners, the author points out that the first experience in the running of the tube mills brought out the fact that with too small a feed the flints and liners begin to grind each other, and the life of the liners increased as the tonnage entering the mill increased. The mills, therefore, are kept loaded to their full capacity, and a tube mill, 13 feet long, grinding to a slime, constantly has a load of close on to 400 tons passing through it, containing about 40 to 45 per cent water. The difference in the wear of the liners between the Rand and Kalgoorlie is thus explained by the fact that in the first place the ore on the Rand is more abrasive than the Kalgoorlie ore, while in the second place the sand enters the tube mill in the latter place in a finer state of division and in larger quantities, taking the size of the mill into

consideration. In the case of the grinding pans the same effect appears, that is, the coarse sand cuts away the shoes and dies much quicker than the fine sand. However, when the shoes and dies of a pan, in good order, are found to wear quickly, it is accepted as a sure sign that the pan has its full load, and the pan acts, therefore, exactly opposite as compared with a tube mill, where the rapid wear and tear of the liners shows that the mill is under-loaded.

In a further contribution to the discussion of the paper under consideration, Mr. J. Thomas points out that the method of giving the speed of revolution of a tube mill at so many revolutions per minute is misleading, and he suggests that the speed be expressed in feet per minute at the internal circumference of the shell. It might be possible to vary the speed of the mills by means of slightly conical pulleys, so that the speed could be altered to compensate the wear of the liners. He mentions that on the Lupardsvlei Estate and G. M. Co., small tube mills have been installed, and that the experience with them so far bears out that of the El Oro Co., Mexico; that is to say, that small mills are more efficient per horse-power consumed than large ones.

**The Computation of the Crushing Efficiency of Tube Mills.**—As the use of tube mills increases it becomes of much interest to make comparisons between individual tube mills or groups of these machines. On this account it is very desirable that a method should be devised by which such comparisons might be accurately made. According to Messrs. S. H. Pearce and W. A. Caldecott, in a recent paper before the Chemical, Metallurgical and Mining Society of South Africa, and printed in the September issue of the *Journal* of that Society, the only method hitherto employed is based on the diminution of the + 60-mesh grade and the increase of the — 90-mesh grade, as is shown in the average gradings of the pulp leaving the battery and of the pulp entering the cyanide works. Thus, for instance, the figures in the following table show a difference of 42 per cent:

| Grade. | Battery Pulp. | Final Pulp. | Difference % |
|---|---|---|---|
| + 60 | 32% | 10% | 22 |
| + 60 — 90 | 12 | 14 | .. |
| — 90 | 56 | 76 | 20 |
| | | | —— |
| | | | 42 |

This 42 per cent, multiplied by the tonnage of ore crushed in the battery during the month, is taken as an index of the tons of sand required. The authors call attention to the fact, that, though this method is certainly simple, it has the drawback of making no distinction between the products of batteries using different size screens, and assumes that there is the same work performed in reducing sand grains to pass a 60-mesh screen, whatever their original diameters may have been. They suggest a method for drawing conclusions from a grading analysis, which is based upon Rittinger's theory that the work done in crushing is proportional to the surface exposed by crushing. The matter thus resolves itself into the mathematical determination of the relative surface area of the grains of the pulp before and after passing through the tube mill, on the assumption that the average shapes of the particles are similar, and that the weights of particles of the successive diameters included in each grade are equal.

As for given weights of ore the total surface area of all the particles in each lot varies inversely as their average diameters, the relative surface exposed by each grade may be obtained by dividing its weight by the average diameter of particle in that grade. Thus the problem is one for the determination of the average diameters of the particles in the various grades into which the crushed product is divided by screening. Assuming now that the average diameter of the particles in the gradings would be represented by the arithmetical mean of the apertures, the following result is obtained:

| Grade. | Mean-Diameter. | Before | | After | |
|---|---|---|---|---|---|
| | | Weight. | Relative Surface. | Weight. | Relative Surface. |
| —.024 + .010 | .017 | .32 | 18.8 | .10 | 5.9 |
| —.010 + .006 | .008 | .12 | 15.0 | .14 | 17.5 |
| —.006 + .000 | .003 | .56 | 186.7 | .76 | 253.3 |
| | | 1.00 | 220.5 | 1.00 | 276.7 |

The additional relative surface exposed is thus found to be 56.2 or 25.4 per cent. The figures given under the heading "grade" refer to the apertures of the screen used, viz.: .024 inch, .010 inch and .006 inch. The authors remark that the defect in the above simple method is that the arithmetical mean of the particles in a grade is greater than the true average diameter, the latter being actually expressed by the following formula of E. Laschinger:

$$d = \frac{d_1 - d_2}{2.3026 \log \dfrac{d_1}{d_2}}$$

where $d$ = the average diameter of the particles in the grade, $d_1$ the maximum, and $d_2$ the minimum diameter of these particles. When this formula is applied to the foregoing case, assuming that the smallest particles in the crushed product have a diameter of .0001 inch, there appears the following result:

| Grade. | Average Diameter. | Before | | After | |
|---|---|---|---|---|---|
| | | Weight. | Relative Surface. | Weight. | Relative Surface. |
| —.024 + .010 | .01595 | .32 | 20.1 | .10 | 6.3 |
| —.010 + .006 | .00783 | .12 | 15.3 | .14 | 17.9 |
| —.006 + .000 | .00144 | .56 | 388.6 | .76 | 527.4 |
| | | 1.00 | 424.0 | 1.00 | 551.6 |

The additional relative surface exposed thus would figure up 127.6 or 30.0 per cent. If the above formula were adopted, a table might be constructed of average diameters between a series of maximum and minimum apertures expressed in thousands of an inch, which would simplify calculation.

**Milling Plants of Summit County, Colorado.**—An article in *Mining Reporter*, Sept. 6, deals with the status of the mills in the above county. It is stated that in the county there are upwards of twenty-five milling plants, having an aggregate capacity of 1,500 tons. More than half of these plants treat the ore by amalgamation and concentration. At several mills amalgamation or concentration alone is practiced, and at one plant, the Masontown mill at Frisco, cyanidation. At the amalgamation plants, stamp crushing is almost universally followed, the stamps varying in weight from 750 to 1,000 pounds. They drop anywhere from 5 to 7½ inches at the rate of 70 to 100 drops per minute. The battery screens have an average of 20-mesh; the plates are as a rule of the normal Colorado type, of an average length of 8 or 10 feet and 4 to 6 feet wide. Inside mortar amalgamation was used at a few of the earlier constructed mills. At one plant crushing mills are used, and at several plants Huntington and Chilean mills are set up. The stamp duty is from 2½ to 3½ tons per stamp per 24 hours. The Masontown mill at Frisco treats the ore by stamp crushing in weak cyanide solution followed by amalgamation, fairly successful results being obtained. Stamp crushing predominates in the amalgamation concentration plants, although at several plants roller mills have been installed, and at one, which is now under construction, a Lane mill will be tried. Regrinding is stated to have been attempted in rare instances, and with indifferent success. The straight concentration plants are stated to employ roll or stamp crushing, the former mainly for quartz-sulphide ores, and the latter for lead and lead-zinc ores. Jig concentration is adopted in the lead and lead-zinc plants, and table concentration at all the plants; the tables include about all known makes, the Wilfleys predominating.

Magnetic separation in the treatment of the zinc ores is stated thus far to have been simply experimental, it being probable that separators will be installed in several mills now in operation or under immediate or contemplated erection. Coarse crushing is the quite general practice on account of the sliming character of the lead ores. It is stated that in the plants using only amalgamation in treating the more free-milling quartz, a recovery of from 60 to 85 per cent of the gold values and 50 to 75 per cent of the silver values has been obtained on ores of value ranging from $10 to $40 per ton. The amalgamation and concentration mills make an average recovery of from 65 to 85 per cent, and the concentration plants recover about 80 per cent. Higher savings are, however, reported in all three classes of mills in several instances.

**Present Milling Conditions in the Cripple Creek District, Col.**—The present state of cyanidation in the Cripple Creek district is reviewed by G. E. Wolcott in the Sept. 6 issue of the *Mining Reporter*. According to him the progress in the local treatment of low-grade ores in this district during the past year by cyanidation without roasting has not in the main been very encouraging. Of the four mills which were actively engaged in the treatment of such ores by the cyanide process, none are now running. Without going into details of the different mills, he feels safe in saying that the majority of failures may be classified as due to two main causes, or rather to two different classes of people, namely, the unscrupulous promoter or the misinformed and unqualified enthusiast. As far as the treatment of low-grade Cripple Creek ores on a large scale is concerned, he feels safe in saying, that it will never be satisfactorily accomplished by simple cyanidation without roasting, or, for that matter, even by the aid of roasting. The Wishbone Co. is seeking a solution of the problem by following the Black Hills practice of amalgamation with subsequent cyanidation of the tailings. The author thinks, however, that while this may give better extraction by the saving of coarse gold, it is doubtful if the process will find any application to tellurides. Roasting, while converting the tellurides into free gold, practically amenable to cyanide treatment, will unavoidably, in the case of Cripple Creek ores, produce more or less coarse gold, which is practically impossible to save by the process. It would seem, therefore, to the author, to follow that the only method of cyanidation applicable to all classes of Cripple Creek ores must be one which includes in the process both roasting and amalgamation, and this necessarily precludes the possibility of the general treatment of low-grade ores. He thinks that recent events bearing upon the handling of Cripple Creek ores seem destined to bring about radical changes in milling conditions, and it is a matter of surprise that no consideration has been previously taken by Cripple Creek operators, looking to the treatment of their own ores.

**The Use of the Filter Press for Clarifying Solutions.**—A paper read before the Chemical, Metallurgical and Mining Society of South Africa, abstracted in the October issue of Electrochemical and Metallurgical Industry, by S. J. Truscott and A. Yates, in which they pointed out the advantages realized by the adoption of the filter press for clarifying gold-bearing cyanide solutions in a plant in Sumatra, called out a contribution from Mr. J. A. Jones, in which he describes the advantages of the use of filter presses, as ascertained at the new plant of the Meyer Charlton Co., on the Rand. When this plant was started two clarifying tanks, each 20 feet in diameter by 5 feet deep, were installed. These tanks had the usual filter bed with mat and jute, covered by 14 inches of coarse sand. It was soon found that their filtering capacity was too small for the requirements and the labor and time wasted in cleaning off the layer of accumlated slime was considerable. One tank, therefore, was removed and an old Johnson press installed which had thirty-four cells, 30 inches square and 1 inch deep, with a filtering area of 425 square feet. In the plant under consideration, cyanide solution is in use

throughout, and the aim is to keep down the gold value of the return mill solution. Therefore, the press was connected with the return launder by a 3-inch pipe, and with a head of 24 feet 500 tons of solution can be clarified daily and passed to the extractor boxes. The press is sufficiently high for a truck to be run under, and is cleaned out about every ten days. A spare set of cloths is then put on, so that the old ones can be washed out at leisure, and in about 2 hours the press is ready to start again. The cloths generally used are a good quality of unbleached calico. The press has a filtering area of 425 feet, but it only occupies a space 14 feet x 4 feet x 3 feet, where, as a clarifying tank, to have the same filtering area, would require a diameter of 23 feet.

The author has also carried out filter press clarification in another instance. The final wash in his large Dehne press is 10 minutes with water, the latter being obtained from the mine. Previously this water gravitated from a reservoir through a 3-inch pipe to a settling tank, and the suction of the washing pump entered the tank about 18 inches from the bottom. A large amount of red slimy ooze, however, accumulated behind the high pressure cloths, and in time interfered with the efficient washing and closing of the presses.

The author interposed one of the earliest patterns of Johnson presses between the reservoir and the settling tank, reducing the connection to 1¼ inches. With only twelve 24 x 1-inch cells this press gives, with 20-foot head, 150 tons of perfectly clear water daily, about three times more than the requirements, and the trouble with the accumulation behind the high pressure cloths has disappeared. In the author's opinion presses are certainly ahead of tanks for the clarification of liquids, inasmuch as, assuming that the first cost of a tank and of a press of a certain filtering area are equal, the labor and time in cleaning the press is much less; the clarifying capacity with even a 15-foot head is much greater; the space occupied by the press is only one-fourth that of the tank, and the solution contained in the press is only about one-sixtieth. This, in the case of gold solutions, is worth consideration.

**Reduction Plant of the Meyer & Charlton G. M. Co., Transvaal.**—The equipment of this plant, which has been erected by the Messrs. Denny, and embodies the latest ideas in the sliming of gold ores and the cyanide treatment of the slimes has been previously described. The following data are given in a note in the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, September, 1906, after the plant has been in operation for ten and one-half months. The stamp duty is stated to be 6.2 tons per stamp when using a 400-mesh screen, the stamps, nominally of 1,100 pounds, but having a net running weight of 1,050 pounds, dropping ninety-eight times 8½ inches per minute, the discharge being maintained as nearly as possible at 2 inches. The theoretical extraction during the above-mentioned period is stated to have been 94.2 per cent, and the actual recovery 94.8 per cent, the slight difference being probably due to the screen assay being a little too low, owing to the system in use.

**Masonry Foundations for Mortars of Stamp Mills.**—The matter of battery foundations built up from wooden blocks versus masonry foundations, has called forth a good deal of discussion, and there still exists much difference of opinion on the subject.

Mr. A. B. Foote, in the *Engineering and Mining Journal*, Nov. 10, makes a strong plea for the masonry foundation. He considers that because some of them, through faulty construction, have proved to be failures, there is no reason for considering masonry unsuitable for motor foundations. He holds that the old-time wooden built-up blocks have been the weakest point in the construction of stamp mills, as they required to be renewed in from eight to ten years at considerable expense, meanwhile causing a serious loss of profit during the time of renewal. In support of his opinion of the masonry foundations he quotes three instances where they have given good satisfac-

tion. In one case of reconstruction of a forty-stamp mill, masonry foundations were built up from the old foundations, and granite capstones, 6 inches wider than the bottoms of the mortars and 18 inches thick, were placed on the top of the masonry, the capstones being long enough to make a close joint under the centers of the battery posts. The latter were cut off and bolted to cast iron pedestals, which in turn were anchored to the masonry, the anchor bolts passing through holes drilled in the granite capstones. The old anchor bolts were used for holding down the mortars, also passing through holes drilled in the capstones. A thin sheet of lead was placed between the bottom of the mortars and pedestals and the capstones. When the mill was started some of the mortars "teetered" on their foundations, and it was finally ascertained that they were slightly convex on the bottom. The tops of the capstones were then dressed slightly concave, and a sheet of ¼-inch rubber substituted for the sheet lead under all the mortars and pedestals. No further trouble has developed, the anchor bolts do not work loose, and the rubber does not seem to wear. The mill has now 1,000-pound stamps dropping 8 inches ninety-six times per minute.

In a second case the mortar foundations of a ten-stamp mill were of concrete, in the proportion of 1 cement, 5 sand and 9 broken stone, nothing being placed between the mortars and the concrete to act as cushion. The top of this foundation went to pieces soon after the mill was started, but after removing the poor concrete to a depth of 2 feet, and building up again with the proportions of 1 cement, 2 sand and 3 broken stone, no further trouble was experienced.

The third case is that of a forty-stamp mill equipped with 1,000-pound stamps, dropping 8 inches ninety-six times per minute, which has now been running over two years. The mortars and the steel battery frame were designed especially for masonry foundations, the latter being of good quality rubble masonry, the top portion being higher grade, with mortar of the proportions 1 cement to 2 sand. The top surface was leveled off with great care, a sheet of rubber 1/32 inch thick being placed between the iron and the masonry. Not the slightest movement of any of the mortars or pedestals has been observed. The conclusions to be drawn are, therefore, that the masonry must be of good quality and the top must fit the bottom of the mortar properly. The more uneven the surface, the thicker must be the rubber cushion, but it is best to have a smooth surface and a thin rubber. The use of anvil blocks under the mortars is not to be recommended, as there is apt to be trouble in keeping the mortars from dancing around on top of the anvils. The author has not observed and does not believe that solid foundations increase the breakage of stems.

**Metallurgical Plants and Practices of Yavapai County, Arizona.**—An article in the *Mining Reporter*, Nov. 1, 1906, deals with the metallurgical methods in the above region, the aggregate daily capacity of the plants being over 6,500 tons, the seventy mills having a total capacity of 3,250 tons, while the six smelting plants have a combined capacity of 3,300 tons. The treatment at nearly all the mills consists in wet concentration, usually preceded by amalgamation, and in a few instances followed by cyanidation. It is stated that of the cyanide plants only at those where the concentrates or tailings material is subjected to roasting before delivery to the leaching vats has cyanidation treatment been attended with any marked success, the Congress plant, of 300 tons capacity, practicing successfully stamp crushing to about 30-mesh, table concentration, roasting of concentrates and tailings, and leaching in a strong solution of potassium cyanide. At the several other plants which successfully practice cyanidation, the slimes and tailings alone after roasting are cyanided, the concentrates carrying precious metals constituting the bulk of the shipping product. The stamps, of which there are over 700 in the county, weigh from 800 to 1,150 pounds, the tendency being of late to set up stamps of 950 pounds or more. They drop generally pretty rapidly, viz.: 100 to 105 drops per minute, of from 6 to 8 inches, the

battery screens having from 20 to 40-mesh, according to the fineness of the gold contained in the quartz. Inside mortar amalgamation is stated to be almost entirely done away with, narrow straight-backed mortars being generally preferred. The amalgamating plates are, as a rule, of a length of from 8 to 12 feet, with widths of 6 feet. At several mills the plates, of a drop pattern, are as much as 18 and even 24 feet long. The duty per stamp is 3 to 4 tons, the recovery by amalgamation ranging from 30 per cent to 75 per cent on oxidized ores, while one plant claims to get a recovery by amalgamation alone of nearly 90 per cent on $15 and $20 highly oxidized ores. On sulphides the recovery by amalgamation is from 10 to 30 per cent. The ratio of concentration varies widely, the average being somewhere between 6 into 1 and 9 into 1. The Pfau mine, in the Cherry Creek section, follows the practice of amalgamation-concentration-cyanidation, the table tailings being reground in tube mills to a fineness of 120 and 160-mesh before being subjected to cyanidation. Wherever regrinding of the coarse table tailings has been done in roller mills of the ordinary type, the recovery by cyanide is stated to have rarely exceeded 75 per cent, whereas the saving at the Pfau plant is reported to have been between 85 and 90 per cent, in continuous operations of several months. The new mills are also equipped with improved mechanisms for a closer classification of battery pulp before it is being distributed to the concentrating machines, a necessity for successful work which was much neglected in the plants built previous to 1900, as it has been in a number of other districts in the West.

### Copper.

**Depreciation of Copper Smelting Plants.**—In a communication to the *Engineering and Mining Journal*, Nov. 10, Mr. E. F. Matthewson, the manager of the New Reduction Works of the Anaconda Copper Mining Co., at Anaconda, Mont., expresses his opinion that the usually accepted depreciation of 10 per cent per annum is too high for modern copper smelting plants. He thinks that such a plant, consisting of the following departments, viz.: concentrating, roasting, reverberatory smelting, blast furnace smelting, converting, steam and electric power plants, should last twenty years, assuming that repairs are kept up in good style. Therefore, only 5 per cent of the cost of the original plant should be written off yearly.

In regard to the question of the salvage obtainable when a plant is dismantled, Mr. Matthewson calls attention to the fact that a great many items of expense will yield no salvage. The principal items of expense are lands, buildings, engineering expenses, interest, machinery and the installation of the same. The values recoverable, for instance, from a typical concentrating and smelting plant for copper, if it were dismantled, are as follows: 44 per cent of the equipment would yield no salvage, 2 per cent would yield 2 per cent of its original cost, 11 per cent would yield 5 per cent of its original cost, 37 per cent would yield 10 per cent of its original cost, 2 per cent would yield 15 per cent of its original cost, 2 per cent would yield 20 per cent of its original cost, and 2 per cent would yield 25 per cent of its original cost; thus, 100 per cent would yield 5.49 per cent.

### Lead and Zinc.

**The Flotation Process at Broken Hill.**—A concise review of the flotation process for the recovery of the zinc and lead sulphides at Broken Hill, Australia, by Mr. D. Clarke, in *Engineering and Mining Journal*, Nov. 24, furnishes some data concerning the present status of these processes. It is stated that in connection with the Potter process, the Goyder Langton plant has been discarded and an ordinary lead-lined spitzkasten, is being used. The tailings are fed into this spitzkasten, most of the zinc sulphide and part of the lead sulphide rising as a thick scum, which continually overflows. The residues fall to the bottom and are continuously discharged. The solution used is sulphuric acid, rarely exceeding 2 per cent of acidity, and it is run in at a temperature near the boiling point, through

a vertical pipe, which discharges the liquid near the bottom of the spitzkasten. It is claimed that from freshly crushed ores over 90 per cent of the zinc contents is obtained. In the Delprat process, while the plant is similar, the spitzkasten has two points, one a blind one called an appendix, and the other the usual discharge opening, the down stream of hot acidulated liquid being directed into the appendix. The solution is said to be common salt added to dilute sulphuric acid. The capacity of these acid treatment plants is stated to be enormous, the plant at the Proprietary mine being erected to treat 1,000 tons per week, while it easily treats double that quantity. The total cost of the plant is very low, as it is simple and easily worked, the feeding of the ore, the discharge of ore and concentrates and the washing of them being done automatically on Robins belt conveyors. The third process in operation, the De Bavay process, depends upon the cleaning of the particles of sulphides with acid, so that when the gangue and sulphides after this treatment are immersed in water, only the metallic substances will float. After the preliminary cleansing with acid, the powdered ore or tailings are fed upon an inclined belt, which gradually delivers them below the surface of a sheet of water; the gangue is then submerged, but a film of sulphides floats and is continuously drawn off. A fourth process, the granulation process, is stated to do good work at the Central mine on slimes; it consists in agitating tailings with a very small proportion of oil and dilute sulphuric acid, at about 100° F. After thorough mixing, the product passes over spitzkasten, the sulphides flowing over the top while the gangue is withdrawn below. The sulphides are cleansed from oil by washing with dilute caustic soda, the saponified solution being allowed to work on fresh ore, the oil being again liberated by the addition of more acid. Of the various theories which have been elaborated to explain the flotation processes, the latest one is promulgated by Mr. De Bavay, who holds that the presence of both colloidal silicic acid and sulphur is necessary for the process to be worked successfully. He claims that colloidal silicic acid is generated when ores containing fluorides are treated with acids, hydrofluoric acid forming silicon fluoride, which in aqueous solution decomposes into silicic acid and hydrofluosilicic acid. Sulphur is liberated from polysulphides. Both of these form colloidal mediums for holding the gas which is evolved, and which attaches itself to the sulphides, and when once a layer or scum of sulphides forms this medium binds them together, preventing them from falling back. Thus any gases evolved will push the layer from below and not drag up individual particles of sulphides.

### Iron and Steel.

**Granulation of Liquid Blast Furnace Slag.**—The question of the efficient granulation of blast furnace slag is of considerable commercial importance, f. i., for the manufacture of slag cement. A new type of apparatus, which depends upon the use of a horizontally arranged cylinder, is described by Mr. G. Hofer in the *Giesserei Zeitung*, Sept 15. The apparatus is illustrated in Fig. 2, and consists essentially of a cylinder *d*, which is provided in its circumference with a series of ribs *c*. The liquid blast furnace slag is conducted to the cylinder by means of a runner *a*, and the granulation can be regulated by adjusting the relative position of the runner *a* with regard to the circumference of the horizontal cylinder, as well as by varying the opening of the runner or regulating the speed of revolution of the cylinder. The ribs *c* are preferably arranged somewhat ascending in relation to the periphery of the cylinder, so that the ribs do not come into contact all at once with the molten slag for their entire length. The lime and other ingredients required for the manufacture of slag cement can be mixed with the slag at the time of granulation, and they can be introduced by means of a runner *b*. This runner is preferably located directly above the slag runner, so that the materials are to a certain degree preheated by the heat of the slag. A construction of this description embodies

various advantages over the type of apparatus which is provided with a vertical shaft and a horizontal circular plate, by which the slag is thrown towards all sides. In an apparatus of this kind the circumferential speed of this horizontal plate at various points of its area is, of course, irregular, according as the point where the molten slag strikes it is more or less distant from the center. Those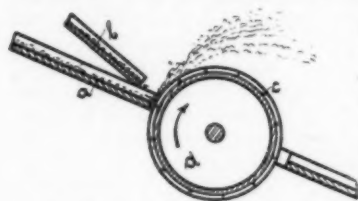 particles of slag which strike more towards the center, are thrown off with much less force and are apt to remain lumpy, instead of being thoroughly granulated. On the other hand, the circumferential speed of the apparatus described above is the same at all parts of the circumference of the cylinder. Moreover, with the latter type of apparatus it is comparatively easy to increase its capacity, which is done by simply lengthening the cylinder, while the widening of a horizontal plate of necessity is confined to very narrow limits on account of the resulting strain upon the vertical shaft and the consequent irregular, wavy motion of the apparatus. As a blast furnace plant usually possesses several blast furnaces, the apparatus can be so installed and the tapping of the furnaces so regulated, that a continuous operation is possible..

FIG. 2.—PULVERIZATION OF SLAG.

# RECENT METALLURGICAL PATENTS.

## IRON AND STEEL.

**Nickel-Silicon Steel.**—A. de Dion and G. Bouten (836,567, Nov. 20) patent the composition of nickel-silicon steels with a high elastic limit and a great resistance to shocks, and said to be especially useful for shafts, gearings, springs, etc. Nickel steels are divided into three classes, according to their microstructure, whether pearlitic or martensitic or polyhedrous. If to a nickel steel of pearlite microstructure silicon is added in the proportion from 0.5 to 2 per cent, with respect to the steel, the microstructure remains pearlite, but the breaking load and the elastic limit are greatly increased, hardly without a diminution of the striction, the lengthening and the resistance to shocks. The following results were obtained with steels annealed at 900° C., and slowly cooled:

|        |      |        | Breaking Load. | Elastic Limit. |
|--------|------|--------|----------------|----------------|
| 0.15 C | 2 Ni | 0 Si   | 41 kg          | 30 kg          |
| 0.15 C | 2 Ni | 1.5 Si | 90 kg          | 65 kg          |
| 0.80 C | 2 Ni | 0 Si   | 90 kg          | 45 kg          |
| 0.80 C | 2 Ni | 1.5 Si | 125 kg         | 75 kg          |

If to a steel of martensitic microstructure 0.5 to 3 per cent of silicon is added, the microstructure is not changed, but the breaking load and elastic limit are increased even more than in the above examples, the fragility of the steel being not increased.

**Brick from Fines and Dust.**—Kondo (836,586, Nov. 20) brings sulphide-ore fines and dust and smalls of coke into brick form for treatment in the blast furnace by pouring the fines together with a stream of molten matte into a mould. The smalls and the matte combine as a conglomerate mass and are later broken up into lumps of proper size.

## SODIUM

**Sodium for Electric Conductors.**—According to E. A. Ashcroft, the present annual production of sodium in the world is about 3,500 tons; of these 1,500 tons are used for cyanide making, and 1,500 tons for peroxide making. In this country metallic sodium is made at Niagara Falls by the Niagara Electrochemical Co., which is controlled by the Roessler & Hasslacher Chemical Co. This company makes sodium peroxide and other peroxides at Niagara, and cyanide at Perth Amboy. The increase of the plant of the Niagara Electrochemical Co. in recent years was necessitated by the increasing demand both of cyanide and peroxides. On the other hand, it would seem that any possible further decrease in the price of metallic sodium might render it available for use in other fields, notably as a strong chemical reagent for various purposes.

The very original idea of using metallic sodium for electric conductors is revealed in a patent of A. G. Betts (833,-290, Oct. 16). It is based on the low specific gravity of sodium and its comparatively good electric conductivity. Weight for weight, sodium has the highest electric conductivity of all metals. To use it as conductor it must, of course, be protected against air; it must be hermetically enclosed in a sheating of another metal. Betts forms a composite conductor by running sodium, while in a molten or plastic condition, into iron or steel tubes of convenient length. The greatest trouble in practice would probably be in making the joints. Betts describes several methods. One way is by plugs, which hermetically seal the ends; another way by sleeve-couplings, in which case the ends of the sodium rods in the two tubes are brought into close contact by heating the pipe until the sodium is fused and runs together. The following comparison is made with copper conductors: "A 2-inch wrought iron pipe has a tensile strength of about 37,500 pounds. Filled with sodium the total weight per foot is about 3.6 pounds. This would carry a current of, say, 860 amps., of which the iron carries about 70 amps. A soft, pure copper conductor of equivalent conductivity is about 1.15 square inches in cross-sectional area, weighs 4.4 pounds per foot, and has a tensile strength of about 32,000 pounds. For conductors of large current-carrying capacity the cost of the composite conductor, even at the present cost of sodium, is very much less than that of a copper conductor of equal capacity." The present price of sodium is, according to price lists, about 50 cents per pound. But that is the retail price. For large, steady contracts it is said that the price goes down to 25 cents per pound.

## TIN.

**Detinning Tin Scraps with Chlorine.**—We have repeatedly referred in these columns to the development of the chlorine detinning process by Dr. Karl Goldschmidt, of the old electrolytic detinning firm of Theodor Goldschmidt, of Essen-Ruhr, Germany. The new chlorine process in its simplest form was described in our October issue, page 419. A patent recently granted to Dr. Karl Goldschmidt and his chief chemist, Dr. Joseph Weber (836,496, Nov. 20), relates to an apparently small but particularly interesting detail of the process. It is essential to carry out the process with dry chlorine, and in such a way that as far as possible anhydrous chloride of tin is produced. In such a dry process the use of washing with water, as disclosed in the present patent, seems peculiar. The reason for the washing is as follows: If bundles of tin scrap have been detinned by means of chlorine or electrolytically with an acid solution, they have a bright steel-gray surface, but after a brief time they rust and are then of no value for treatment in the open-hearth furnace. It has been considered to be one of the special advantages of electrolytic detinning in an alkaline solution that the bright surface of the iron produced thereby does not corrode. Goldschmidt and Weber have now found that the corrosion referred to before in connection with the chlorine process is due to a very thin but firmly adhering layer of chloride of iron. They state that the chlorine treatment completely removes the layer of tin and does not attack the iron. To explain the formation

of the chloride of iron, they have recourse to the very small interconnecting layer between the actual sheet iron and the actual tin, and which may be a kind of alloy of iron and tin. This alloy is said to be attacked by the chlorine, with the result of formation of chloride of iron. If this is left on the surface of the iron the latter will rust, but rusting can be completely avoided by a washing process with water, the chloride of iron being dissolved; and by subsequently flushing the iron sheets in a weak alkaline bath a thin protective layer is formed, which prevents any corrosion.

**Detinning Tin Cans.**—A second patent of the same inventors (836,028, Nov. 13) is perhaps still more interesting from a commercial point of view. The detinning industry has already grown to considerable dimensions, but it is probably not generally understood that so far practically nothing but the scraps from tin-can factories have been detinned. The enormous amount of tin cans which have been in actual use was not available for successful economical treatment up to the present. The reason is to be found in the necessity of getting pure tin and pure iron as end products. Used tin cans are full of impurities, and anybody familiar with the practical necessity of maintaining

an electrolytic solution pure will appreciate the difficulties which would be experienced in the electrolytic detinning of used cans. In this problem everything is to be considered as impure that is not either iron or tin; not only the rests of preserves (caviar, herring sauce, etc.) which were contained in the cans, but also the paper labels, the lacquer, the coloring matter, the solder and the thin caoutchouc ring which in folding the edges of the tins is placed in the fold to make the same tight. To remove all these impurities, Goldschmidt and Weber employ a process of two steps, the first of which destroys all adhering organic substances, not including the above-mentioned caoutchouc strips. This is done by dipping the cans in a hot 3 per cent solution of sodium hydroxide. The cans are then washed with water, and in the second step of the process the solder and the caoutchouc strips are removed by sudden exposure of the boxes to a temperature of from 600° to 800° C. The solder runs off and is recovered, while at the place of the caoutchouc strips there are narrow channels which will allow the detinning substance to circulate. The tin boxes are then ready for detinning. It is reported that this process is now being successfully used to some extent in the Goldschmidt works in Essen.

# ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

### ELECTRIC FURNACES.

**Soft Graphite.**—E. G. Acheson, 836,355, Nov. 20. Application filed Sept. 22, 1906.

In our September issue (page 343) we noticed that Mr. Acheson has succeeded in finding a simple way for making soft graphite in the electric furnace. The artificial graphite heretofore made by the International Acheson Graphite Co. is hard, and is well suited for use for electrodes for electric furnaces and electrolytic cells, as a paint, pigment, etc. Soft graphite will be useful especially as a lubricant, a stove polish, and for electrotyping and other purposes. The pure, soft and unctuous graphite produced by the new method does not coalesce under pressure. The process consists in "heating in an electric furnace a charge comprising essentially carbonaceous material, such as mineral coal, coke, petroleum-coke and the like, and one or more carbide-forming material, the latter being in excess of the proportion of natural ash contained in any coal, but in less than the theoretical proportion required for production of a carbide, that is, the carbon is present in greater proportion than is required to reduce the metallic compound or carbide-forming material and to combine with the base thereof with formation of a carbide." The following specific example of the process is given. An electric furnace, having a length of 18 feet between terminal electrodes, was provided with a starting core consisting of a graphite rod ⅞ inch in diameter. The active zone, 18 inches in diameter, surrounding this core was filled with a mixture of carbonaceous material and carbide-forming oxide. The materials used in this specific instance were anthracite coal, ground to pass through a ½-inch mesh, mixed with sand, in the proportion of 65 per cent coal and 35 per cent sand, the ash contained in the coal being calculated as a part of the sand content of the mixture. Completely surrounding the active zone above referred to was disposed a mixture of anthracite coal and sand in the proportion of 1 part coal to 2 parts of sand, this mixture having a much higher resistance than that in the active zone, and serving as an effective heat retainer. The furnace being charged in this manner the electric current was turned on, and at the beginning registered 79 volts and 75 kilowatts. After 2 hours the register showed 203 volts and 200 kilowatts, and after 9½ hours showed 135 volts and 800 kilowatts. The register at the of 15 hours still showed 800 kilowatts, while the volts had dropped to 70,

as the result of decreased internal resistance, due to the formation of graphite. When cold the furnace was opened and 962 pounds of soft, unctuous and non-coalescing graphite were removed from the active zone.

Silica is preferred by the inventor as carbide-forming material, for the reason that oxides which form fluid carbides are more or less difficult of treatment in an electric furnace.

**Terminals for Electric Furnaces.**—F. J. Tone, 836,155, Nov. 20. Application filed Nov. 11, 1905.

To get a good contact between the end of the carbon terminals and the metallic clamping bars (which make connection with the external circuit) a layer of compressed artificial graphite powder is interposed between metal and carbon. The metallic clamping bars are cored out to provide channels for the circulation of cooling water. To protect the exposed portions of the carbon terminals from oxidation, they are surrounded with a layer of cemented refractory material, for which purpose various silico-carbides, such as crystalline and amorphous carborundum and siloxicon are found to be well adapted. This refractory material is mixed with a suitable binder, such as cement or silicate of soda, and the mixture is brought to the consistency of stiff mortar. These refractory jackets may also be provided with circulating ducts for cooling water.

**Plastic Cylinder of Fused Silica.**—J. F. Bottomley and A. Paget, 836,558, Nov. 20. Application filed Feb. 28, 1906.

A cylinder of fused silica is produced in an electric furnace containing a carbon or graphite rod as resistor, around which glassmaker's sand is placed. In order to produce an externally glazed product the rough cylinder is subsequently treated in a separate furnace, or the glassmaker's sand is contained in a jacket of carbon or platinum, which is further protected on the outside against heat losses by additional sand or other heat insulating materials. The jacket is made in two parts so as to be easily removed. The resistance core is also made removable. With respect to the further treatment of the plastic cylinder of fused silica when removed from the furnace, instructions for drawing the cylinder, for blowing it and for compressing it between rollers or dies are given.

**Furnace Construction.**—Frank J. Tone, 834,948, Nov. 6. Application filed Sept. 14, 1905.

The object is to increase the economy of operating electric

furnaces which have large dimensions in the direction of the path of the current, the current being introduced into the furnace through terminals at the two ends, and passing from end to end through a conducting core or through the charge.

To make the working more convenient, the furnace walls are made of removable and transferable sections, and the bottom of the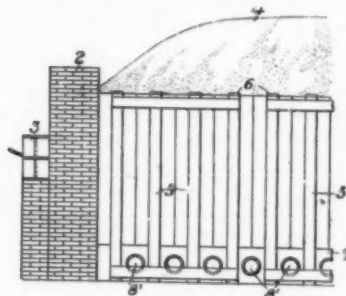 furnace is artificially cooled, since otherwise at the existing high temperature the masonry composing the bottom becomes conductive, and current is shunted off through it and is wasted. One form of construction is shown in Fig. I, giving a side view of part of the furnace; 2 is a permanent end wall supporting the electrode 3, while 4 is the furnace charge. The sections of the side walls are composed of bricks assembled in iron frames 5, and held therein by retaining lips 6 at top and bottom; 8 are popes in the bottom through which cooling water is passed.



FIG. I.—ELECTRIC FURNACE.

**Molybdenum.**—Fred. M. Becket; 835,052, Nov. 6. Application filed Feb. 1, 1906.

The object is to produce fused molybdenum, low in carbon, directly from the sulfide ore, molybdenite, in a single operation. For this purpose, a mixture of molybdenite, carbon and an oxide or other compound of an alkali or alkaline-earth metal is treated in an electric furnace. The reactions resulting if lime is used or calcium carbonate, are, respectively,

$$2MoS_2 + 2CaO + 3C = Mo + 2CaS + CS_2 + 2CO$$

and

$$2MoS_2 + 2CaCO_3 + 5C = 2Mo + 2CaS + CS_2 + 6CO$$
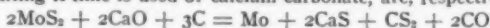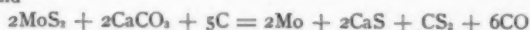
A modification of the method is to treat a mixture of molybdenite and a carbide of an alkaline metal such as calcium carbide, the reaction being in this case

$$5MoS_2 + 2CaC_2 = 5Mo + 2CaS + 4CS_2$$
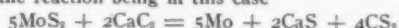
The advantage of the process is that the produced molybdenum has a low carbon contact (not more than 0.2 per cent), and is free from molybdenum oxides; further that is produced from molybdenite in a single operation and with cheap reagents. For the production of ferro-molybdenum or a molybdenum-nickel or other alloys, the inventor introduces, of course, metallic iron, nickel, etc., into the charge. The process has an apparent outward likeness to the Huntington-Heberlein lead process, but in reality lime plays an essentially different part in this process (see our vol. III., pages 363 and 454) than in the above reactions. The latter have, however, many analogies with the work of Brown and Oesterle (see our vol. III., page 378).

**Carbon Tetrachloride and Calcium Carbide.**—J. M. Matthews, 835,307, Nov. 6. Application filed Jan. 26, 1904.

A mixture of calcium chloride and coke is heated in an electric furnace for the purpose of producing carbon tetrachloride and calcium carbide, according to the equation

$$2CaCl_2 + 5C = 2CaC_2 + CCl_4$$
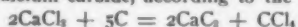
The well-known corresponding equation of the ordinary calcium carbide process is

$$CaO + 2C = CaC_2 + CO$$

Calcium chloride may be had cheaply, since it is a by-product of various chemical industries. On the other hand, carbon tetrachloride is a valuable product, having come into considerable favor as a volatile solvent of a non-inflammable and non-explosive character.

The carbon tetrachloride is conducted through cooled con-densers, and thus brought to liquid form. It is necessary that the electric furnace reaction should take place in an atmosphere free from any oxidizing or decomposing influence. Moreover, in order to obtain carbon tetrachloride free from lower chlorides of carbon, the condensation of the highly heated vapors should take place in an atmosphere of chlorine. The electric furnace illustrated in the patent is of the well-known rotary calcium carbide furnace type, but is provided with a closed flue for carrying off the carbon tetrachloride vapors, and with an inlet pipe for the introduction of non-oxidizing gas. In the flue between furnace and condenser a chamber is interposed for condensing other volatile compounds that may be carried over from the furnace. Into this chamber also a current of chlorine gas is introduced.

**Electric Furnace.**—E. A. Story, Oct. 2. Application filed Sept. 13, 1905.

The furnace is designed primarily for fusing refractory substances, such as vitrifying enamel, etc. Fig. 2 shows a cross-section. The brick 21 to be fused rests on a little car running through the furnace chamber. There are two rods 13 passing through the left side-wall of the furnace (there being only one shown, the other one being behind it). These two rods are connected together by a bonded conductor 16. Through the right-hand side-wall of the furnace passes a corresponding pair of carbon rods 14, also bonded together by 16. A slab 17 of thoria, magnesia, or kaolin, which is substantially a non-conductor when cold, but becomes conducting with increasing temperature, forms the resistor which is directly
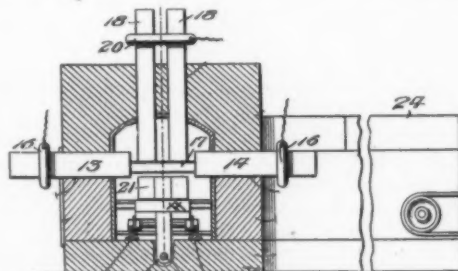


FIG. 2.—ELECTRIC FURNACE.

over the brick 21 to be fused. An additional pair of carbon rods 18 are vertically movable through the top of the furnace and are bonded together by means of 20. The electrodes 18 rest on the resistor 17. The electrodes 13 and 14, together with the resistor 17 form one conductor, while the electrodes 18 constitute the other conductor. At the rear end of the furnace a tunnel or hot-air chamber 24 is provided, which extends at a right angle to the length of the furnace. An endless conveyor receives the vitrifying material from the trucks in the furnace chamber and carries it to a point of exit.

**Treatment of Zinc-Lead Ores.**—F. T. Snyder, 834,644, Oct. 30. Application filed June 21, 1905.

The inventor has proposed to treat zinc-lead ores in electric furnaces from which air is excluded, the ore being smelted with carbon in the furnace to reduce both the zinc and lead, the lead being collected in liquid form, while the zinc is volatilized and condensed. Under these circumstances he found that a part of the lead also volatilized and was carried over with the zinc vapor into the condensers. He now provides an electric furnace through which the ore is passed gradually from the colder to the hotter end, all the lead being melted and carried away at the colder end, and pure zinc being volatilized at the hotter end. The furnace is shown in Fig. 3, where a is the electric smelting chamber. The temperature along this chamber is graduated by maintaining a body of slag of varying depths within it and passing the heating electric current through the slag. Carbon electrodes *f f* are provided at the upper end of the smelting chamber, while the lead col-

lects in the well $e$ at the other end and serves as the other electrode. This well $e$ communicates with the exterior of the furnace by U-shaped passages. The current passes through the slag from $f$ to $e$. The ore is introduced through the charging tubes $l\,k$ and preheated by means of burning fuel in the furnace chamber м. This is done to start the process of reduction as far as possible and to economize in this way the more costly electric heat required in the smelting operation. The limit of preheating is that slag must not be formed in the charging tubes, since this would quickly destroy them. Since the cross-section of the body of slag is greater at the end near $e$ that at the end near $f$, and since the slag at the
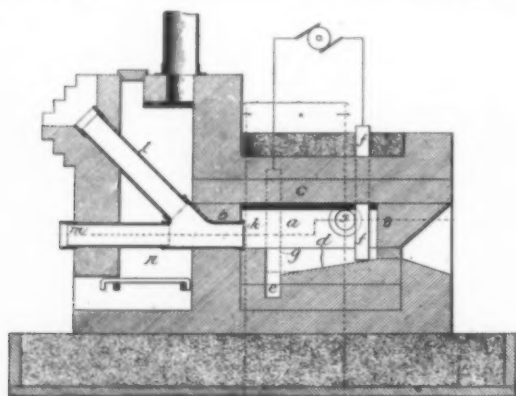


FIG. 3.—TREATMENT OF ZINC-LEAD ORE.

top is of higher resistance than the molten lead in $e$, the greatest heat is developed near the electrodes $f\,f$, where the slag bath is of least depth and will be gradually diminished in intensity toward the end $e$ of the furnace. As the ore and coke are advanced into the smelting chamber $a$, the lead is first reduced and settles out below the body of ore and slag, running down the sloping hearth into the well $e$. The unreduced portions of the ore are then gradually advanced to the hotter parts of the furnace and zinc vapor passes off through the flues $s$ into the condensers. Since the tubes $l$ and $k$ are always filled with material, no appreciable amount of air will enter the furnace during the charging operation, and care must be taken to always exclude air from the furnace. The temperature of the furnace in the hottest part should be about 1200°. The materials of the furnace charge should be proportioned so as to produce a slag which will form a temperature of 1000 or 1100° C. A slag containing approximately 30 per cent of lime, 30 per cent of iron oxide and 40 per cent of silica will have the desired character.

### Electrolytic Processes.

#### Electrolytic Precipitation of Precious Metals.—J. Snodgrass, 835,329, Nov. 6. Application filed Jan. 3, 1906.

This patent, coming from Johannesburg, is interesting, since it shows that work with electrolytic precipitation of gold has not been entirely abandoned in the Transvaal in favor of zinc precipitation. The precipitation box is shown in Fig 4. The solution enters the box through 7 and leaves it through 8; 9 are the anodes, and 10 the cathodes. The latter rest on a sheet 11 of iron on the bottom, which is in electrical connection with all the cathodes. The special feature of the process is the construction of the porous electrodes. The cathodes 10 consist of very fine iron-wire gauze in an iron frame. A screening of 10,000 meshes per square inch is satisfactory. Inside of the gauze a woven fabric of fine texture may be used, which is rendered a conductor by a coating of plumbago. The conductivity may be increased by dipping the cloth into a solution of a lead salt and then into a solution of an alkaline carbonate or sulfate. By this means

a deposit of carbonate or sulfate of lead is formed on the fibres of the cloth, and after the cathodes have been placed in position in the box, the first passage of the current reduces the lead to the metallic state, and so increases the conductivity. A combination of cloth and wire gauze may be used, woven fabrics being placed on both sides of the wire gauze network. The anodes are constructed with a wooden frame with pieces of porous cloth of very open texture, such as flax scrim. The compartments thus formed are filled with an insoluble conductor in the form of powder or small lumps of gas carbon or coke. The solution flows slowly through the box, and when passing through the porous cathodes the precious metals are deposited on the same. The iron plate 11 at the bottom prevents the redissolution of any particles of
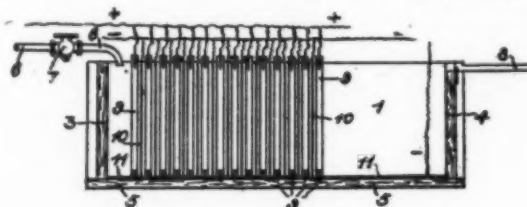


FIG. 4.—GOLD PRECIPITATION.

gold which may become detached from the cathodes 10 and come in contact with the plate 11.

#### Electrolytic Production of Bases of Alkali-Earth Metals.—A Brochet and G. Ranson, 835,661, Nov. 13. Application filed Feb. 14, 1901.

The operation consists in the electrolysis of a mixture of the sulfide of an alkaline earth with a small quantity of a chloride. Under these circumstances, hydrate of the alkaline earth is formed at the cathode, and sulphur is deposited at the anode without any evolution of chlorine. The following equations represent the reduction:

$$SrS + 2H_2O = Sr\,(OH)_2 + H_2 + S$$
$$BaS + 2H_2O = Ba(OH)_2 + H_2 + S$$
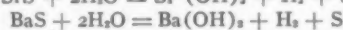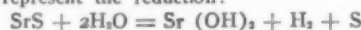
When a diaphragm cell is used, the sulfide of the alkaline earth is placed in the anode compartment, and the alkali chloride in the cathode compartment. The chloride is made to flow in a continuous manner into the apparatus, and becomes charged with the base of the alkaline earth, and on passing out is led into the crystallizing apparatus in which the base of the alkaline earth becomes deposited by cooling. The latter is separated by centrifugal action from the mother liquid, which is returned to the apparatus. The anodic liquid consisting of a mixture of the sulfide of an alkaline earth and of a chloride of an alkali, may be when needed conducted from the apparatus and filtered, reheated and subjected to systematic lixiviation in the presence of the sulfide of an alkaline earth, whereby it may be kept saturated. The use of the chloride as catholyte enables one to obtain the base of the alkaline earth in a pure state, that is free from sulfide.

#### Extraction of Gold.—W. A. Hendryx, 836,380, Nov. 20. Application filed Sept. 20, 1904.

The mill is arranged on a hillside, so that the ore and solution are automatically transferred by gravity from one step of the process to the next one. At the top the ore supply and the stock-solution tank are provided. The latter is filled with a weak cyanide solution. The ore, after being treated with rock breakers, is subjected to wet crushing in mortars, and then passed over an amalgamating table to recover any free particles of gold or silver. The cyanide ore-pulp solution is then run into a Chilian mill and is reground. It is then run over another amalgamating table and passed into settling and classifying tanks, where suitable chemicals are added, which thoroughly eliminate any deleterious acid salts the raw pulp may contain, and which would interfere with the full action of the cyanide solution. The ore pulp flows from one compart-

ment of the tank to another, and the pulp is separated from the cyanide and the cyanide is pumped back to the stock solution tank at the top of the mill. The ore pulp as it settles from the solution is divided and classified into different sizes by gravity, from the coarsest particles in the ore pulp to the finest slime. The heaviest particles settle in the first compartment, and the next largest in the second, and so on to the finest and lightest slime, which will be the last to settle. The several sizes are then drawn from the various compartments in sufficient quantity to make a charge of desired quantity and consistency of ore pulp in the agitating tank. This is the next step of the process, and represents its most important feature. The construction of the agitating tank will be described some-
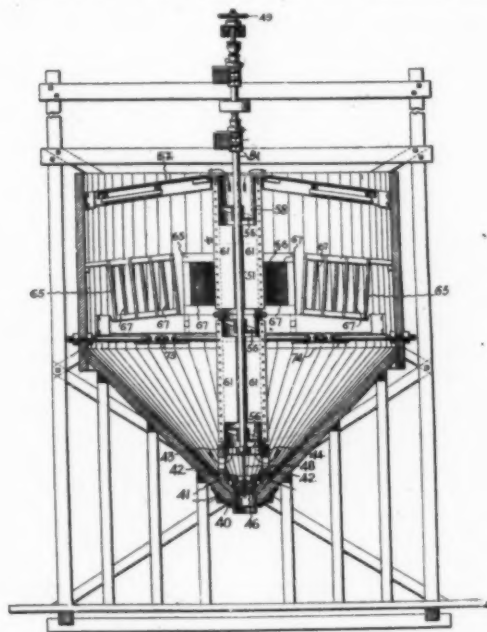


FIG. 5.—EXTRACTION OF GOLD.

what more in detail below. In this agitating tank the solution pulp is circulated, aerated, and the gold is deposited by electrolysis. After treatment in the agitating tank the ore pulp solution is passed into settling tanks. The clear solution is returned by pumping up to the stock-solution tank, while the settled ore pulp is discharged.

Fig. 5 shows the agitating tank with the discharge aperture 40 at the bottom, surrounded by the valve seat 41. The upward projecting legs 42, at substantially equal distances apart, are connected at the top by a flat ring 43, in which is secured the spider 44; 48 is a stem by means of which the valve 46 can be opened and closed with the aid of hand-wheel 49. This valve stem is surrounded closely but loosely by a tubular shaft 51, carrying three screw propeller blades. The screw propellers cause circulation of the solution, as indicated by the arrows in the illustration. The solution pulp discharged at the top flows over the conical deflector plate 62 in a thin film, whereupon it drops downwards and is subjected to electrolysis between a system of anodes and cathodes. At four symmetrical places in the solution, electrode-supporting grames 65 are provided, which contain carbon anodes 66 and lead cathodes 67. These are so placed as to stand at an angle so that the surfaces of the cathodes are in the direct path of the ore pulp as it falls from the peripheral edge of the deflector into the tank.

**Electrolytic Production of White Lead.**—E. D. Chaplin, 836,177, Nov. 20. Application filed Feb. 3, 1906.

The inventor uses an electrolytic cell of three compartments, the electrolyte, which consists of a mixture of sodium nitrate and sodium chloride, being supplied to the middle compartment, so as to prevent any mixture of the products formed at the anode and at the cathode. The cathodes are of copper, while the anodes are of lead, which goes into solution. In the cathode compartment sodium hydrate is formed, and in the anode compartment lead oxychloride; the latter being an acid salt renders impossible the formation of insoluble basic salts of lead. By a reaction between the lead oxychloride with sodium hydrate in a tank outside the electrolytic cell, lead hydrate is then produced. Some of the sodium hydrate is treated with carbon dioxide, with the result that sodium bicarbonate is formed. By reaction between the lead hydrate and the sodium bicarbonate in another tank, lead carbonate is finally produced. By means of this indirect process it is claimed that a white lead of color-carrying and covering capacity is produced, which is greatly superior to that obtained by the direct process in which the soluble acid salt of lead is directly carbonated. The process is continuous, and, in addition to the lead and carbon dioxide consumed, requires only the addition from time to time of fresh quantities of neutral solution.

**Electrolytic Production of Hypochlorites.**—R. Kother, 832,-983, Oct. 9. Application filed Nov. 3, 1905.

Mechanical details of construction of an electrolytic cell for making bleaching liquors. The construction is concisely summed up in the third claim, which reads as follows: "Electrolytic apparatus for the manufacture of bleaching liquor comprising compartments separated by non-conducting (vertical) partitions, a single electrode arranged horizontally in each non-conducting partition and projecting on either side thereof in such manner that each two adjoining half-electrodes lie horizontally one above the other forming a step-like arrangement, of the electrodes, the upper projecting half of an electrode in each compartment being perforated, the lower electrode portion in each compartment acting as an anode and the projecting half of the next electrode situated above it acting as a cathode."

**Treating Cotton and Wool.**—G. E. Burton, 827,293, July 31. Application filed Nov. 20, 1905.

In a vat containing an aqueous solution of sodium carbonate and sodium chloride (in the proportion of 2 to 1), with specific gravity 1.02, stirred and heated to about 80° F., wool or cotton fabrics are first exposed to electrolysis. The result is stated to be an opening, expansion and strengthening of the fibers of the wool, together with removal of the grease. The wool thereby gets a soft and velvet feeling. The solution is then drawn off into a separate vat and allowed to settle, and then brought back into the first vessel in which now an equal quantity of raw cotton is treated electrolytically for 15 or 20 minutes. The result is that the cotton takes on the appearance of wool, absorbing a certain amount of the wool grease and animal matter (left in the solution from the first operation), and giving to the cotton an appearance and feeling of wool.

**Galvanizing Wire.**—G. L. Meaker, 830,093, Sept. 4. Application filed June 18, 1902.

To economize space in an electrolytic tank for galvanizing wire, drums are provided at the top and the bottom of the tank, and the wire is passed over them so as to form loops. Within each loop an anode is provided.

**Electroplating Cylindrical Articles.**—R. C. Totten, 827,473, July 31. Application filed Jan. 31, 1905.

The object is to electroplate circular or cylindrical bodies, such as metal working rolls, treads of car wheels, pulleys, tires, reels, journals, tubes, etc. The patent refers essentially to mechanical details of a support on which the article to be plated is slowly and uniformly rotated.

**Anode.**—J. Nelson, 830,918, Sept. 11. Application filed June 28, 1905.

In order to use metal in granular, lump or scrap form for anodes in electroplating vats, the inventor uses "an anode,

comprising a plate equipped with vertical marginal cleats, and a porous envelope comprising a sheet stretched across the cleated face of said plate and having its lower margin folded under the lower edge of the plate and its lateral margins folded and laced together."

### Apparatus for Electroplating.—A. J. Leaver, 835,960, Nov. 13. Application filed June 11, 1906.

To agitate and circulate the plating solution, the inventor employs a screw propeller in a trunk immersed in the solution and extended as nearly to the bottom of the vat as may be desired. The trunk is bent through a right angle, the horizontal branch of it in which the propeller is placed being normally near the surface of the solution. Means are provided by which its depth can be varied. If it is desired to aerate the solution, the trunk is raised until the propeller is partly above the surface.

### Electrolytic Production of Sodium.—C. F. Carrier, Jr., 830,051, Sept. 4. Application filed Jan. 30, 1905.

The construction of the cell is shown in Fig. 6, with the anode compartment in the center in form of a rectangular frame 4, 5, with graphite anodes 9. On each side of the anode compartment a cathode compartment is provided in form of a rectangular box or bell 13, containing the iron cathode 14, the lower surface of which is slightly inclined upwardly from the sides towards the center and towards one end. The middle portion of the bottom of the electrolytic cell (opposite the anode) is raised somewhat above the level of the two end portions (opposite the cathodes). The electrolyte 17 in the two cathode compartments is fused sodium hydroxide. The elec-
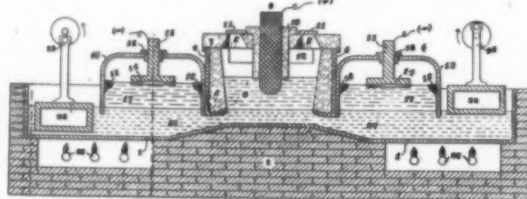


FIG. 6.—PRODUCTION OF SODIUM.

trolyte 6 in the anode compartment is fused sodium chloride, while below the electrolytes in these compartments a layer of fused lead 27 is provided which acts as a bipolar electrode and prevents the electrolytes in the anode and cathode compartments from mixing. The chemical reactions are exactly the same as in the Ashcroft process. In the anodic compartment chlorine is set free at the anode and conveyed away through the port 12, while sodium is set free at the lead 27 and alloys with it. When this lead-sodium alloy has been transferred to the cathodic compartments 17 the sodium passes into the sodium-hydroxide electrolyte and metallic sodium is set free at the cathodes 14 and is recovered. The sodium-hydroxide is not consumed. To circulate the lead-sodium alloy between the anodic and cathodic compartments, the two plungers 22 are put into a reciprocating motion, as is at once understood. In the cathode-compartment bells 13 an atmosphere of nitrogen or hydrogen is provided to prevent any oxidation of the metallic sodium which is set free. To prevent any short-circuiting between the cathode 14 and the bell 13, the cooling pipes 18 are provided, which extend entirely around the bell just below the surface of the electrolyte. In this way the electrolyte just around the cooling pipe is solidified and made a non-conductor.

### BATTERIES.

### Storage-Battery Plate.—J. H. Robertson, 835,229, Nov. 6. Application filed Feb. 1, 1904.

By means of special tools, a series of horizontal oppositely communicating recesses are cut into the plate, "each being formed in the arc of a circle, whereby finely tapering parts are formed between the ends of the recesses and at each side of a central aperture, the bottom surface of each recess being longitudinally grooved." The object is to provide a large active surface for plate formation.

### Storage Battery.—P. Schmitt and C. Fabre, 835,642, Nov. 13. Application filed Nov. 6, 1902.

A frame is made of strips of land somewhat in the form of a book-shelf, and the front and back are closed by perforated walls of celluloid or wood. The partition in this frame are filled with active material, and the special feature of the construction is that the active material is in the form of grains lying one on the other without being bound or soldered together. The horizontal sheets on which these grains rest act as conductors for the current.

### Storage Battery.—E. W. Jungner, 836,261, Nov. 20. Application filed Sept. 11, 1905.

The patent refers to the manufacture of plates for his storage battery, which, in its chemical nature, is essentially the same as the Edison nickel-iron cell. The method comprises "feeding continuous metallic ribbons toward each other, perforating the ribbons, then feeding an active mass between said ribbons and finally uniting the ribbons and severing the electrodes thus formed."

### Storage Battery.—T. A. Edison, 827,297, July 21. Application filed July 21, 1904.

In order to avoid the presence of active sulfides in his alkaline storage battery, vulcanized rubber insulators or supports are used, wherein all free sulphur capable of going into solution is preliminarily eliminated.

### Primary Battery.—I. Kitsee, 827,914, 827,915, 827,917, Aug. 7. Application filed June 12, 1901, Feb. 20, 1904.

The inventor endeavors to avoid the use of a depolarizer in a primary cell, and tries to achieve this by using a platinized cathode which will absorb any hydrogen developed. His cathode consists of a rubber sheet on which rests a porous or perforated carbon plate, which is covered and filled with an "active material" consisting of platinized charcoal. The patent 827,917 refers to a combination of a number of primary cells in the form of a battery, whereby all the cells are connected to two reservoirs, one containing the electrolyte and the second the depolarizer, and means are provided for emptying and cleaning the cells.

### Storage Battery.—W. Gardiner, 827,861, Aug. 7. Application filed Sept. 23, 1904.

The two plates are provided above one another. The lower plate is in form of a wire net, resting on the bottom of the cell, while the upper plate is made of two closely perforated metal sheets, the space between the two sheets being filled with active material. The construction seems to be specially intended for alkaline cells.

### Storage Battery.—W. Gardiner, 827,968, Aug. 7. Application filed Aug. 31, 1904.

One electrode is formed of a sheet of pure lead closely perforated and folded transversely to provide a plurality of closely-arranged folds which are jammed close together, forming horizontal pockets adapted to contain active material. This plate is electrochemically formed. A silver-plated copper wire net on both sides of and below the lead plate forms the other electrode. The electrolyte is a solution of sulfuric acid and water containing sulfate of zinc, sulfate of lithium and bisulfate of mercury. In charging the metal constituents of the salts are deposited on the wire net, and constitute the negative pole-element, the wire screen serving but as a support for the deposit.

### Dry Cell.—Angelica E. Post, 828,335, Aug. 14. Application filed May 17, 1905.

The ingredients are compounded in an absolutely dry condition and packed in the cell in such a way that they maintain their dry condition until required for use. Moisture is then introduced.

**Galvanic Cell.**—B. Jonas, 828,319, Aug. 14. Application filed April 2, 1904.

The cell contains horizontal carbon plates surrounded by granulated carbon, forming one electrode. The other electrode is granulated iron. Diaphragms are provided in form of porous cups and canvass walls. Mechanical details are provided by means of which the cell is periodically filled and emptied "with electrolytic fluid, which periodically drives out and fills the cell with air, which very greatly assists the cell in its generating power."

**Separator for Storage Batteries.**—L. H. Flanders, 836,107, Nov. 20. Application filed April 15, 1905. Assigned to Westinghouse Machine Co.

To separate the plates from each other, encircling insulating rings are used, provided with a downwardly-extending lug, arranged to coöperate with a recess formed in the electrode.

**Dry Cell.**—G. L. Tarver, 836,151, Nov. 20. Application filed March 5, 1906.

A cup of zinc sheet is amalgamated on the inside and lined with pasteboard paper. The carbon electrode in the center rests on the paper lining, and has packed around it a bottom layer of dry plaster of paris. On top of this is a thicker layer of silicious sand, and above this comes a mixture of powdered carbon and manganese dioxide. At the top is another layer of sand. The carbon-manganese dioxide mixture is brought to the consistency of a paste by adding a sufficient amount of solution, which consists of 3 parts by weight of water with 1 part salammoniac, combined with a cold mixture of 5 parts water and 2 sulphuric acid.

**Dry Cell.**—J. W. Brown, 836,480, Nov. 20. Application filed Sept. 14, 1906. Assigned to National Carbon Co.

To get a uniform internal resistance and thereby a uniform current density at all parts of a dry cell, the inventor makes the diameter of th ecylindrical zinc can larger at the top than at the bottom, and the carbon electrode thinner at the top than at the bottom. In this way the distance between the carbon electrode and zinc can is gradually shortened from the top downwards to compensate for the increasing distance along the carbon electrode from the binding post.

### DISCHARGES THROUGH GASES.

**Ozonizer.**—M. Otto, 827,387, July 31, 1906. Application filed July 1, 1905. Assigned to American Ozone Co., Niagara Falls, N. Y.

Details of construction of an ozonizer, the elements of which are readily changeable. The particular feature is a metallic electrode forming an in-draft chamber for the air, and placed between two glass dielectrics, each of which is covered with a conducting layer connected with the ground. Several methods for producing the in-draft of the air are described. It is said that the apparatus should include in practice a cold-air generator. "Liquid air enriched by oxygen at a low temperature gives very good results."

**Ozonizer.**—A. Déchaux, 830,975, Sept. 11. Application filed April 28, 1904.

A battery of ozonizing units, each of which consists of three tubes. The outer and inner tubes are of metal, the middle one of a dielectrics. The three tubes are fitted within each other with slight friction. The outer and inner metallic tubes may have polygonal cross-section, while the middle glass tube may be cylindrical.

## Notes on Electrochemistry and Metallurgy in Great Britain.

*(From Our Special Correspondent.)*

THE POWER PROBLEM IN RELATION TO BRITISH ELECTRO-CHEMISTRY AND ELECTROMETALLURGY.

Mr. W. B. Esson, in delivering his presidential address to the Civil and Mechanical Engineers' Society, dealt in an inter-esting way with the question of the supply of power to factories in general, taking as his basis a factory equipped with an engine giving 500 hp. coupled to an electrical generator working at about two-thirds load during 2,800 hours, or 1,000,000-e. h. p. hours. Taking London prices for fuel and allowing 10 per cent for interest on capital outlay, the total costs work out as follows:[1]

|  | Mond Gas. | Suction Gas. | Steam (con.) | Steam (non-con.) | Oil. |
|---|---|---|---|---|---|
| Works cost | 0.226d. | 0.299d. | 0.255d. | 0.284d. | 0.280d. |
| Capital charges | 0.161d. | 0.124d. | 0.130d. | 0.118d. | 0.164d. |
| Total cost | 0.387d. | 0.423d. | 0.385d. | 0.402d. | 0.444d. |

Of course, the above figures are for a very high load factor, much larger than that of any works, other than those devoted to electrochemistry and electrometallurgy, with which I am acquainted. Mr. Esson, however, draws a fair inference when he states that it will pay the large consumer better to use power generated at his own works instead of purchasing from a supply company.

One reservation should have been made in regard to the hours of the consumer's demand. If the industrial economy of his works permit it (in other than in cities stricken with sudden fogs), so that his power load would go off as the peak of the lighting load came on, the large consumer could undoubtedly get cheaper power from the supply company. The small consumer scores all along the line, and at 3 farthings and at 1 penny per unit, which are common charges in London, the isolated plant cannot possibly compete.

London can never be a large chemical or metallurgical center. The electroplating industries also must tend to center round Birmingham, where the works to which electroplating is subordinate are situated; electrometallurgical industries of the electric furnace type must be associated with works at the source of one or more of the component raw materials; electrochemical industries concerned in the preparation of electrolytic alkali and bleach must center around the Cheshire and Middlesborough salt districts.

Electrochemical and electrometallurgical processes of the intermittent type would probably obtain their power cheaper if it could be arranged that their operation was discontinued for 2 hours in each 24 during the peak of a lighting load. Where this is impossible the separate generating plant is absolutely essential. There is enough power going to waste in the gases of our blast furnaces at present used for firing boilers to supply power at a rate slightly cheaper even than that estimated by Mr. Esson.

### DR. HUTTON'S LECTURE ON THE ELECTRIC FURNACE.

The provincial engineering societies frequently begin their Winter session of papers before the London societies. The first paper of the Winter dealing with matters of interest to your readers was Dr. Hutton's missionary journey to Sheffield, to the local Engineering and Metallurgical Society, to read a paper upon "The Application of the Electric Furnace to the Metallurgy of Iron and Steel." One uses the word missionary advisedly, for it was a remarkable fact that in a paper presented to the Iron and Steel Institute, at their Autumn meeting in Sheffield last year, concerning the equipment of the Metallurgical Laboratories at Sheffield University, no mention was made of an electric furnace as part of the equipment.

Dr. Hutton's main thesis was to the effect that the advanced state of British mechanical engineering put us in the first position in regard to the possibility of power generation by steam, and that under conditions such as prevail for industries in which the electric furnace is used electrical energy could be obtained at 0.25 pence per unit in favorable localities and with

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[1] It is very questionable whether the figures of Mr. Esson will be accepted on this side of the water. That steam power can be cheaper than gas power will certainly be most strongly disputed.—Editor.

large units. Taking this as his basis, Dr. Hutton proceeded to urge that the British steel maker should refrain from scoffing at new processes, just because they tried to do more than had been previously possible. Instead of propounding arguments and objections, which only served to show that the new process had not been properly understood, it would be worth while to go very thoroughly into the details of the whole question.

The cost of electricity was, after all, only a small fraction of the total costs, that even with steam power the proposition was not at all unfavorable. Sheffield was still the home of the highest class steel making, and British manufacturers had such an intimate acquaintance with the intricate technicalities of the subject, that they should be able to make the electric furnace go considerably further than those who had started with only a hazy knowledge of steel.

### The Institution of Mining and Metallurgy.

The first ordinary meeting of this institution was held on Oct. 18, when three papers were discussed. The first by Mr. Malcolm Maclaren on "The Auriferous Rocks of India, Western Australia and South Africa," was mainly geological, the conclusions being set out in the following table and notes:

*India.*

Cambrian or Pre-Cambrian....Karnul beds.
                     Cuddapah (Kadapa) beds.
                     (Gwaliors and Bijawars of North India).

Archæan .....................Dharwar series (Aravalli, Chota, Nagpur and Shillong of North India).
                     Fundamental gneissic granite.

*West Australia.*
Oakover beds.
Nullagine beds.

"Auriferous series."
Fundamental gneissic granite.

*South Africa.*
Witwatersrand system.

Swaziland schists.
Bulawayo schists.
Namaqualand schists. Malmesbury series.
Gneissic granite.

Briefly summarizing, it may be stated emphatically for India and Western Australia, less so for South Africa, that: (a) in all three areas two distinct periods of auriferous deposition may be made out. (b) Both periods are connected with an original basic igneous rock, the older period occurring during the metamorphism of the igneous rock into a schist, the younger period being subsequent to the penetration of intrusive diabasic rock through the schists so formed. During the latter period the schists were mineralized anew, and, as in Western Australia and South Africa, overlying sedimentary rocks received some portion of the gold circulating in solution. (c) The younger periods may broadly be regarded as contemporaneous, and their basic eruptions as a Palæozoic prototype of the world-wide tertiary effusion of andesites with accompanying auriferous solutions.

The second paper, on "Sand Sampling in Cyanide Works," by Mr. Duncan Simpson, commenced by pointing out that "Important as is the sampling of the untreated and treated sand (respectively designated charge and residue) to the control of the recovery of gold in a cyanide works, consistent results seem to have been obtained in many cases more by accident than design. Carelessly-taken samples, at the end of a sufficiently long period, may give an average figure closely approximating to the true gold value, but instances are not unknown, more commonly when the slime separated from the mill pulp was run away to dams, of the figures representing the tonnage capacity of the treatment tanks being altered to bring the gold recovery within the limits of what has been considered normal extraction; making the tonnage treated a variable quantity dependent on the value of the sand." Eight methods of sampling were, therefore, described and discussed. Of these the author, speaking from his experience at the New Goch Gold Mine, described as unreliable the method of taking a sample by catching a sample from the delivery hose at regular intervals, and also the use of an automatic sampler. The only samples at present being taken are at the mill screens, at the discharged tailings belt, and from the slime filter presses, together with an occasional check by applying No. 4 method, in which half the sand was discharged from the tanks, leaving at the diameter a vertical face 7 feet high, which was divided into seven horizontal zones, each 1 foot wide; the face of each zone was scraped into a bucket, and the seven samples thus obtained were dealt with individually and collectively.

The third paper, on the "Treatment of the Precipitate and Manipulation of the Tilting Furnaces at the Redjang Lebong Mine, Sumatra," by Mr. S. J. Truscott, does not lend itself to abstracting in these notes, on account of the concise manner in which it is written.

### An Annealing Furnace for Tool Makers.

One feature of an electrical exhibition held at Sheffield this month was an electric furnace for annealing and hardening, which was exhibited by the Electrical Co. (the English branch of the A. E. G.). Its essential features consist of two wrought iron electrodes in a rectangular crucible, a bath of a fused electrolyte of barium chloride or barium and chloride of lime, and a regulating transformer step-down transformer having a range of from 2 to 50 volts. Tests with a pyrometer indicate that the temperature is equable throughout. The following interesting table gives the energy consumption per cubic inch of electrolyte for a medium sized furnace to maintain a constant temperature:

750° C. is about 4.10 watts.      1,000° C. is about 23,000 watts.
800° C. is about 7.05 watts.      1.150° C. is about 36.10 watts.
850° C. is about 9.85 watts.      1,300° C. is about 49.20 watts.

Of course upon starting there is a heavy flow of current for a few minutes until the electrolyte becomes fluid.

### Improved Conditions in the Electrolytic Alkali Trade.

The annual report of the director of the Electrolytic Alkali Co., Ltd. (who work under the Hargreaves-Bird patents) for the year ended Aug. 31 last, states that the net profit, after allowing for depreciation, mortgage debenture interest and expenditure on renewals, repairs, up-keep of buildings, plant, etc., is £12,095, added to £528 brought forward, making £12,623. Three years' dividend on the preference capital having now accrued, the directors propose to pay one and a half years' dividend upon these shares, which will absorb £10,496, leaving £2,127 to be carried forward. The favorable result is attributable to increased output and reduced cost of production.

### Inventions, Inventors and the New Patent Act.

A paper read by Mr. E. Dunbar Kilburn before the Junior Institution of Engineers of Oct. 16, on the "Protection of Inventions," was mainly devoted to an attack upon the law which came into force on Jan. 1, 1905, which was chiefly notable for the introduction of an official search into British Patent Office administration. Speaking as a patent agent the author regarded the act as having been followed by a disillusionment of the popular mind, and that the inventor is not benefiting financially and otherwise as had been expected. The chief point urged is that as the official search is only undertaken upon the filing of the completed specification, it will not compare in utility with a search upon the filing of the preliminary specifica-

tion. One result may, indeed, be the disappearance of the preliminary specification.

Turning now to another question in connection with the British patent practice which had recently come in for a good deal of discussion, namely, as to whether it should be made compulsory for a holder of a British patent to manufacture in Great Britain the article protected, the author strongly held the view that the compulsory working of patents is wrong. The general tendency abroad, in those countries where the law renders working compulsory, is towards relaxation of the requirements, and it may be that at no distant date it will be found that it has become obsolete. Again, theory is opposed to practice. It sounded very nice to say that if a man owns a patent in a country he must manufacture in that country the articles covered by the patent, or at any rate a substantial proportion of what he sells in the country, in order to benefit local industries. In practice, however, it frequently happened that those who are struggling to start an invention found the continual drain on their pockets, in consequence of the recurring need for proving workings on their foreign patents, in excess of their income in the early stages, with the result that they let these patents lapse and leave the establishing of the industry or the introduction of the article into that country to take care of itself. If, on the other hand, they had been free to import articles manufactured in the home country, and so start the market for the new article, they would probably in the end have found a purchaser who would establish a local factory, and the ultimate result would be what the present legal requirement has in view. No doubt in the case of England there is much importation of patented goods made abroad, but the reason in many cases will probably be found to be either that the expenses of manufacture in England are much greater than in the inventor's home country, or that the slowness, which is so unfortunately apparent in many English firms to take up a new article and risk something in putting it on the market, drives the inventor to importing and building up a demand on his own account if he wished to get any sale at all outside his home country. Compulsory working, in the author's opinion, got at the question in the wrong way. If they wanted to give the home manufacturer a chance in competition with the foreigner, which is after all what those who advocate compulsory working are aiming at, it would seem that they must do it in a different way, and get at the foreigner when he imports. Let him make abroad as much as he liked, and let him import as much as he liked, but on terms which would prevent him from competing unfairly with the home manufacturer. Also let the facilities for the granting of compulsory licenses be made use of.

### MARKET QUOTATIONS.

Prices are still ruling on the high side, led by the exceptional strength of the metals. A large business was done in copper sulphate, which is now strong at £31.10 per ton, an increase of £1 per ton in the past fortnight. Shellac has reached the high price of 220/ per cwt., and Para rubber is at 5/1½ to 5/2¼ per pound. Copper remains firm at £99. Tin again shows an upward tendency, closing at £196.12.6 cash, and £197.10 three months, having risen £4 and £3 respectively since the end of October. Zinc was £27.10 on Oct. 31. and had fallen slightly by the time of writing (Nov. 6) to £27.5. Lead shows little variation from £19.10, a slightly upward tendency if anything.

LONDON, Nov. 7, 1906.

# BOOK REVIEWS.

THE MINERAL INDUSTRY, VOL. XIV. Edited by W. R. Ingalls, New York: *Engineering and Mining Journal.* Price, $5.

We have received the Mineral Industry for 1905. The general excellence of this series is an enduring monument to Mr. Richard P. Rothwell who founded the annual. The scope has been well carried out by Mr. W. R. Ingalls, editor of the *Engineering and Mining Journal.* By considerable effort the volume has been brought out this time at an earlier date than in the past two or three years. The general excellence of the book is beyond question and it certainly surpasses any other technical annual published in any language. American mining men and metallurgists have a right to be proud of this book as representative of the best in American technology.

It is useless to tabulate the general articles and their respective authors. Among these might be mentioned James Douglas, chief engineer of the Phelps-Dodge copper interests and well known as a graceful writer on general subjects as well as an expert copper smelter. Prof. H. O. Hofman and Prof. Rob. H. Richards, of the Massachusetts Institute of Technology. The fourteen volumes of the Mineral Industry comprise a complete up-to-date encyclopedia of the mining and smelting business, valuable alike to the capitalists, engineer and young student.

We could have wished that the articles on electrochemistry and electrometallurgy had not been dropped, and that short biographies with photographs of the different authors had been inserted as was the custom "Consule Planco." But these are but captious criticisms in such a fine production.

METALLURGY OF CAST IRON. By Thomas D. West. Fully illustrated. Eleventh edition. Cleveland, Ohio: The Cleveland Printing Company. Price, $3.00.

This book is so well known to foundrymen, and its intrinsic value is so well indicated by the appearance of the eleventh edition in less than ten years after the first edition, that little need be said concerning its technical value.

The book is well characterized on the front page as "a complete exposition of the processes involved in the treatment of cast iron, chemically and physically, from the blast furnace through the foundry to the testing machine—a practical compilation of original research."

The first part treats of manufacture and use of coke; properties in ores; operations of blast furnaces; the different brands of pig iron and how to purchase and use them intelligently.

The second part deals with the elements in cast iron and their physical effects; the utility of chemical analyses, and how to use them in making the different mixtures of irons employed in making gray and chilled castings.

In the third part, after discussing the properties of and methods for testing molten iron, the author discloses phenomena in the activities of cooling metal, etc., and presents results of tests in all kinds of iron and best methods for testing.

The book is concluded by a series of selected useful tables for furnace and foundry work.

Aside from the technical value of the book, the eleventh edition, under review, is most interesting as a human document. The chief objects for which the author has been persistently fighting in his long professional life have now been accomplished.

His standardized drillings of cast iron have been taken over by the Bureau of Standards in Washington, and it may surely be hoped that this will tend toward greater uniformity in the work of chemists in making pig iron and the mixing of iron by analysis, for making castings, etc. Much progress has been made in establishing standard methods for making chemical determinations of the different constituents of iron, as well as in grading pig iron by analysis.

These are only a few points for which the author of the book has been successfully contented, and the foundry industry owes a lasting tribute to him for his indefatigable work. We only wish that the book may be as productive of good results in the future as it has been in the past.

THE COPPER HAND-BOOK.—A manual of the copper industry of the world, Vol. VI. Houghton, Mich.: **Horace J. Stevens.** 1,116 pages. Price, $5.00.

This is the sixth annual edition of the well-known Copper Hand-Book. It is to be acknowledged that the author endeavors to keep the volume fully up to date. For the present edition the last two chapters have been entirely rewritten.

Of these the one dealing with the copper mines of the world represents, as heretofore, the most important portion of the book. This chapter now comprises 916 pages, and gives concise information on about 5,000 copper mining companies. While the information given is concise and to the point, there are occasionally delightful touches of humor. Thus in the glossary of mining terms an apex is defined as "that part of an ore vein at or nearest surface; usually requires opposing experts and several lawsuits to determine; in case of litigation the apex is usually owned by the litigant having the most money." In speaking of the Trinity Copper Co., the company of Mr. Lawson, of Boston, the complete working forces of the concern are summed up as follows: "Forces, one $10,000-a-year-general manager, and two $3.00-a-day watchmen."

The book in its new edition will undoubtedly be welcome to the many friends which it has won in former years.

## Concentration and Briquetting of Iron Ores

We have repeatedly called attention in these columns to the fact that improvements in metallurgical methods often create new commercial values whenever they render possible the economical treatment of materials which formerly were practically valueless.

A most interesting example is presented in the progress made in recent years in concentrating low-grade iron ores and in briquetting the concentrates or fine ores for further treatment.

In the following notes the processes of Mr. Gustaf Gröndal shall be described, which have already proven commercially successful in various plants, notably in Sweden and Norway, and seem liable to find extended application also in this country. The American patents are owned by the American Cröndal-Kjellin Co., 45 Wall Street, N. Y.

### CRUSHING AND CONCENTRATING OF MAGNETIC ORES.

The Gröndal crushing and concentrating process is commercially applicable to poor magnetites containing down to 25 per cent iron as well as magnetites with a high percentage of phosphorus and copper not chemically combined with the iron. The crude ore is reduced to about ½ inch cube in a crusher, and is then treated together with water in a Gröndal ball mill, from which it escapes as pulp in a finely ground condition. Each mill requires 20 to 25 hp. and treats 50 to 100 tons or ore in 24 hours.

The ore thus crushed is then passed through a magnetic separator. In order to get rid of the bulk of the non-magnetic slimes it is often advisable to pass the pulp coming from the ball mill through a slime box before charging it into the separator proper.

This slime box consists of an ordinary V-shaped box receiving a stream of clear water from the bottom. Two of these slime boxes are arranged under a horizontal electro-

magnet, either pole of which terminates in a hatchet-shaped pole piece, the edge of which nearly touches the level of the pulp in the box.

The dimensions of the box and the velocities of the pulp and clear-water currents are so arranged that everything except the finest slimes settles in the box, and passes through a pipe to the magnetic separator, the fine slimes overflowing and being drawn off to waste. Any magnetic matter contained in these slimes is arrested when it comes within the magnetic field produced by the wedge-shaped pole piece and accumulates at the surface of the water until it forms masses of such size as to drop down and be carried away through the above-mentioned pipe by the issuing stream of water; the strength of the electromagnet is such as just not to lift any of the magnetic particles out of the water.

The pulp freed from slime now passes to the separator proper. A magnet with pole pieces of the same shape as those used for the slime boxes is surrounded by a drum composed of alternate bars of soft iron and brass. This drum rotates about 1 inch above the surface of the pulp, which traverses a pyramidal box, this being divided into two compartments by a partition reaching nearly to the top of the box. The pulp enters at the side opposite to the partition and a stream of clear water entering from the bottom and rising up on the same side of the partition, carries all the pulp well over the edge of the latter, and thus immediately under the drum.

The bars of iron composing the latter become powerful magnets so long as they are within the very strong magnetic field of the pole pieces, and they therefore pick up all the magnetic particles from the stream of pulp. The remainder of the pulp drops down on the other side of the partition and is carried off by the stream of water through the waste outlet.

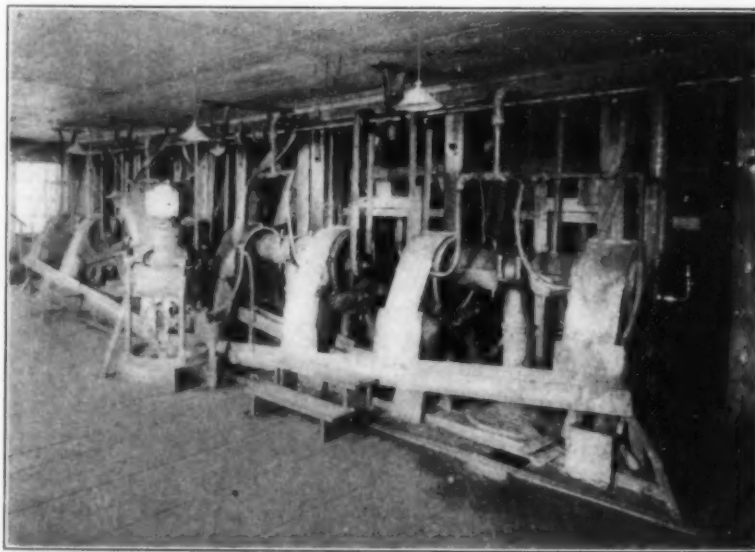The pure magnetite is lifted by the drum to the very edge of



FIG. 1.—GRÖNDAL MAGNETIC SEPARATOR.

the magnetic field, where it is thrown off by the speed at which the drum revolves. The middlings, consisting of particles that are in part magnetite and in part barren rock, are flung off before they reach the weak part of the field. The capacity of such a two-drum separator is about 50 tons in 24 hours; it requires about 6 amperes at 120 volts, or say 1 e. h. p., while less than ¼ e. h. p. will drive the drums.

Some of the results lately obtained from the new type of drum separator at Herrang, give concentrates 60 per cent to 61.5 per cent iron, middlings 7 per cent to 10 per cent iron, and

tailings 5 per cent to 8 per cent iron, and from 70 to 100 tons of ore have been treated per double-drum separator per 24 hours.

### BRIQUETTING.

The Gröndal process of briquetting, which has attracted wide-spread attention, serves to prepare the concentrates obtained by the above described processes for treatment in the blast furnace, but is, of course, also applicable to the treatment of fine ores and sands.

The powdered ore is pressed into briquettes of suitable size without the use of any binding material whatever, the moisture in the powder being so adjusted as to obtain briquettes sufficiently firm to be removed from the press to the cars used in the furnace. These are made with a frame of iron covered with fire-brick. The briquettes are placed on edge on the cars, the number of layers depending on the material to be briquetted.

The loaded cars are transferred one by one to a long tunnel-shaped furnace, which is gas fired, the combustion chamber being situated about the middle of the furnace. All the cars are made with a groove along one end and a projecting rib at the other, and as they are advanced by being pushed in, these fit gas-tight; the longer sides of the cars are fitted with a deep flange, with dips into a channel filled with sand and running the full length of the furnace.

The row of cars thus constitutes an air-tight horizontal partition, below which the air needed for combustion is admitted, keeping the wheels and framework of the cars cool. The row of cars does not reach quite to the discharge end of the furnace, and the air current passes below the cars to that



FIG. 2.—BRIQUETTE PRESS.

end and returns above them towards the combustion chamber, traversing the mass of burnt briquettes, cooling these and at the same time becoming itself heated.

The products of combustion pass over the entering cars of briquettes as these advance towards the combustion chamber, thus heating these and becoming themselves so thoroughly cooled that they escape into the stack at a temperature of under 100° C. The cars of briquettes are drawn at about the same temperature, when a car is taken out regularly every half-hour.

Owing to these applications of the regenerative principle, the thermal efficiency of the furnace is high, the chief source of loss of heat being the evaporation of the water contained in the raw briquette. When briquetting iron ore concentrates, and using producer gas, the consumption of coal in
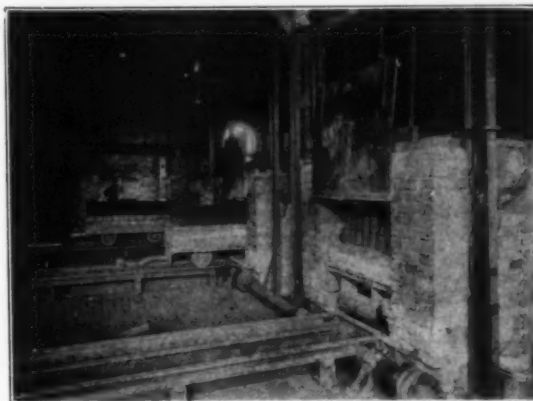


FIG. 3.—BRIQUETTE FURNACE.

the producer has been found to average 7 per cent of the weight of briquettes burnt.

The temperature in the combustion chamber reaches 1,300° or 1,400° C., and at this heat the particles agglutinate sufficiently to form a firm, hard briquette, able to stand rough treatment and long transport. The briquettes, though hard, are very porous, and are consequently far more easily reduced in the blast furnace than ordinary lump ore.

Briquettes made by this process from pyrites residues and purple ore have proved to be entirely satisfactory for use in the open-hearth steel furnace.

The output of one furnace varies from 30 to 100 tons in 24 hours, according to the class of ore used and the degree of desulphurization required, for in addition to its mechanical action, the briquetting furnace acts as an exceptionally efficient calciner for removing practically the whole of the sulphur contained in the pulverized ore, as shown by the following analysis of ore before and after briquetting, the figures representing per cents:

| | | | |
|---|---|---|---|
| Crude ore | 39.3 Fe. | 1.13 S. | 0.006 P. |
| Concentrates | 62.9 | .27 | .003 |
| Refuse | 11.4 | 1.58 | .017 |
| Briquettes | 61.1 | .008 | .003 |
| Pig iron from briquettes | | .005 | .012 |

These results were obtained at Herrang, in Sweden.

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### Recording Types of Le Chatelier Pyrometer.

The Le Chatelier pyrometer, with its platinum rhodio-platinum thermo-couple and indicating galvanometer, has won an enviable position for the thermo-electric pyrometer in general. But the indicating form of pyrometer does not meet the requirements of many superintendents and managers. They want to know what has been going on during the hours they were not at the particular place in the works where the pyrometer is installed. They want also a record to be marked and filed away which will show the exact heat treatment that was given in each particular run, so that it can be duplicated some other time or so that it may be noted and avoided if the product be unsatisfactory. They do not wish to repeat their failures, but they do wish to repeat their successes.

The result has been that there have been enough demands for a good recording pyrometer to enlist the services of the masters of that type of instrument building, and it is not

strange that the most complete and serviceable recording pyrometers have come over to us from Paris, where it has been Le Chatelier himself that has given the stimulus, with the mechanical details and design worked out by the most clever and finished instrument maker in France.

In Fig. 1 is shown the recorder which has its record sheet mounted upon a drum, and each sheet accommodates a record of twenty-six hours' run. The instrument is in a solid milled aluminium case, which, of course, cannot warp. The needle carries a reservoir pen, and the drum, counterbalanced and controlled by clockwork, comes up to meet the pen at short intervals. The record is very distinct, and its variations from a straight line show the changes in temperature. An excellent feature is the very open scale secured by having a width of 4¼ inches for the record. But we shall refer to another form of this recorder, and in connection with it shall speak of some features that are common to both.

This second very beautiful instrument (Fig. 2) is made to look a little ungainly because we illustrate it with the case open so as to show somewhat of the interior. As with the other, the case is of solid aluminium, thick, strong and dust-proof. Projecting down from the galvanometer needle is a stylus. Under the stylus travels a carbon belt, and under that is the record paper. Over the needle there is a beam, which is one piece, with the two long arms supporting it. This beam is raised by clockwork, and is allowed to fall of its own weight once a minute. The needle, which has been free to swing to its proper position, is depressed by the beam, and the stylus causes the carbon belt to make a dot on the record paper.

The record paper is thin, but opaque, and is in the form of a very long ribbon wound upon an aluminium spool. A single spool carries enough of the paper to last a whole year of continuous running, makes this instrument particularly valuable for use where a process of long duration is to be put under control.

The current generated in the thermo-couple, and which it is the duty of the recorder to measure, is the only electric current that has any function in the instrument, as the whole recording operation is controlled by an eight-day clock. By not employing any relay currents all danger of direct or induced changes in the light current to be measured is avoided. Each instrument has two different internal resistance values, either of which may be employed by using the proper binding posts, of which there are three.

In the usual course of operation, the case should be closed and locked. The clock can be wound and started or stopped without lifting the case. The record feeds out of the instrument as does the tape from a stock "ticker," and as there is a new roll of paper needed but once a year, if the instrument is continuously run, it can readily be seen how little chance there is for anyone to tamper with the works.

Of no less importance in the pyrometer is the thermo-couple, whose wires must be homogeneous to avoid generation of parasitic currents along the wires. The Wilson-Maeulen Co., of New York City, who are supplying these recorders, have obtained wires for thermo-elements that are strictly homogeneous, and yet which are sold much more reasonably than couples have been supplied for heretofore that have had certifi-



FIG. 1.—THERMO-ELECTRIC RECORDING PYROMETER.

cates from the national laboratories of the European countries.

The couples this firm supply are tested, when the purchaser desires, by the Bureau of Standards at Washington. We have seen some of the certificates from the Bureau on their couples, which show that though made from wires not contiguous they yet all have the same values in electromotive force at the same temperature.

The firm announce that they can supply wires of which exact duplicates or other quantities having the same values can
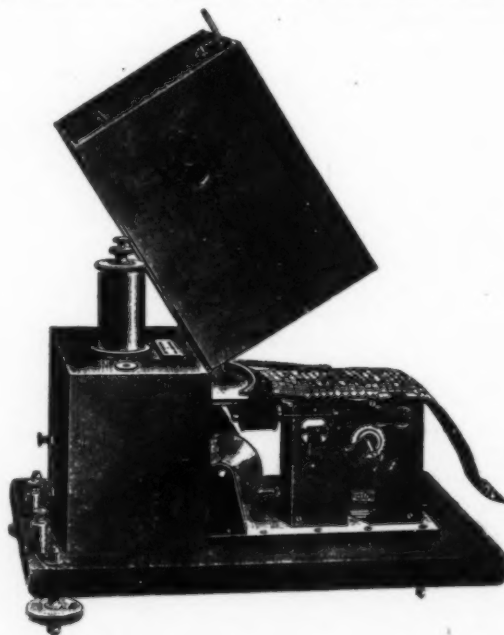


FIG. 2.—LATEST TYPE OF THERMO-ELECTRIC RECORDING PYROMETER.

be supplied for some years to come. The Bureau's report on the homogeniety of the wires shows them to be of the highest quality.

One element is of platinum and the other of rhodio-platinum, but platinum iridio-platinum will be supplied to anyone preferring it, although nowhere but in England does the latter enjoy much favor. It is perfectly reliable up to 900° C., but should not be used above that. When it is understood that platinum and irridium alloy do not hold constant above 900° C., though both metals and all their alloys melt at over 1,700° C., it is not difficult to understand why couples of base metals and their alloys are proving false in their operations when in use over 750° C. The recorders above mentioned are supplied with base metal couples when 750° C. is the upper limit at which readings will be taken.

## Notes.

**Tube Mills in Gold Metallurgy.**—The United States Reduction & Refining Co. of Colorado Springs, have recently installed eight Abbé tube mills, 5 feet by 23 feet, equipped with "Ideal" spiral feed and discharge for regrinding in the cyanide process. Another mill of the same make, measuring 4 feet by 14 feet, has been installed at the plant of the Silver Peak Valcade Gold Mining Co., at Silver Peak, Nev.

**New Publications.**—Within the past fifteen or sixteen months, the Publicity Department of the Allis-Chalmers Co. have covered with new bulletins about 80 per cent of the products manufactured by the company, and it has taken nearly 80 publications, averaging about 24 pages in size, to do so. This is in itself a good illustration of the variety and extent of the products of the Allis-Chalmers Co. When their list of

publications is completed, the number will be considerably in excess of 100. They have recently completed a list of those publications which were in force on Oct. 1.

**Seattle Exposition.**—Much interest is now taken in the West in the Alaska-Yukon-Pacific Exposition, to be held in Seattle in 1909 from June 1 to Oct. 15. The primary purpose of the fair is to exploit the resources and potentialities of Alaska, Yukon and the Pacific Northwest, and to make known and foster the vast importance of the trade of the Pacific Ocean. It is estimated that the fair will cost $10,000,000. The exposition site comprises 255 acres of the campus of Washington University, and the plan of the Exposition includes the erection of permanent buildings, which are to be taken over later by the University for educational purposes.

**The End of the Municipal Hypochlorite Plant in Philadelphia.**—Several years ago we had occasion to visit the small municipal hypochlorite plant in Philadelphia. It was located in an old house not far from the Public Building and was at that time only intermittently operated. It was stated, however, that the hypochlorite solution had proven quite satisfactory for disinfecting houses after contagious diseases. From a technical point of view the construction was quite primitive. The cathodes were made of zinc—a peculiar thing since an attendant might reverse the current. In the recent report of the Chief of the Electrical Bureau, Mr. John C. Sager, to the Mayor of Philadelphia, we find the following note: "The disinfecting plant for the manufacture of electrozone, which had been operated by this Bureau since 1895, having been discontinued during the last administration, was dismantled and brought into the office."

**Pressed and Seamless Drawn Steel Articles.**—We have received from Messrs. Janney, Steinmetz & Co., of Philadelphia, Pa., their illustrated article on pressed and seamless drawn steel specialties, such as tanks, shells, cylinders, bottles for compressed gases, etc. Each article is worked out of circular discs or rectangular pieces of plate by means of a series of operations through dies. The material is carefully annealed and worked at each operation, thereby maintaining the original vitality of the metal. The steel is low in carbon, sulphur and phosphorus, has a high limit of elasticity, and a tensile strength of 55,000 pounds to the square inch. In manufacture the circular discs or plates are pressed into cups of such diameter and length as the finished product demands. The cup is then cold-drawn into a finished product of required size. Any defect in the raw material shows itself in the early drawing operations. Such material is immediately rejected. The cold drawing makes the product smooth inside and out, and greatly increases its tensile strength as well as making it dense and tough. The gradual working of the material makes it possible to produce light and very strong articles, accurate in dimensions, free from seams, welds and riveted joints.

**Gas Power.**—We have received from the Backus Water Motor Co., of Newark, N. J., their catalogue on Backus suction gas producers and gas engines. In this catalogue we find the following comparative operating costs of different engines:

of the American Chemical Society, as the next speaker, presented to Sir William the diploma as honorary member of the American Chemical Society. Further speeches were made by President N. M. Butler, of Columbia University; Rev. Dr. S. P. Cadman (who spoke in an extremely interesting and impressive way); Prof. H. Schumacher, one of the German Exchange professors who is now lecturing at Columbia; Dr. Ira Remsen, of Johns Hopkins; Prof. Walter Nernst, of Berlin; Prof. H. W. Wiley (in his typical humorous style), and Comptroller H. A. Metz. In his reply, Sir William first expressed his sincere thanks to all former speakers, and then gave at length an account of his struggles in his early days, how his father wanted him to become an architect, how he was attracted by chemistry, and became one of the lecture assistants of Thomas Hall in the City of London School, to enter later, at the age of 15, the Royal College of Chemistry in London, where the celebrated W. A. Hoffman was professor of chemistry. There he invented mauve at the age of 18. He described how his father and brother helped him to introduce his invention into the industry, and gave an account of his further work as a manufacturer, which ceased in 1873. After this date Sir William occupied himself purely with scientific research.

**Graphite Electrodes for Electric Furnace Work.**—The great activity in electrochemistry in this country is indicated by the fact that the total amount of graphite electrodes supplied by the International Acheson Graphite Co., of Niagara Falls, for use in electric furnaces and similar work during the year ending July 1, 1906, was 2,404,171 pounds. It is, of course, to be understood that this embraces only the quantity of material supplied in electrode form, since very large quantities of artificial graphite are sold by the Acheson Co. for other purposes, for instance, for paint pigment. It is also known that the consumption of amorphous carbon electrodes is also very large, since for a wide range of electric furnace work the cheaper amorphous carbon electrodes will do equally well. There have been recently some very interesting developments in the manufacture of very large amorphous carbon electrodes by the National Carbon Co., of Cleveland, the electrodes produced being larger than any which have been made before.

**Calendar.**—Messrs. C. W. Leavitt & Co., in New York, importers of foreign ores, metals and alloys for iron, steel and brass works and dealers in chemicals for glass works, have sent us their calendar for 1907, with the picture of an automobile, broken down, and being hauled home by a horse under the guidance of a grinning farmer.

**Applications of Ultraviolet Light.**—By a misprint, which our readers will undoubtedly have recognized as such, the term efflorescent is used instead of fluorescent in the report of Dr. Baskerville's lecture on page 435 of our last issue.

**T. Shriver & Co.**, manufacturers of filter presses, formerly of New York City, announce the removal of their office and works to Harrison, N. J., where they have increased manufacturing facilities and better railroad accommodations for immediate shipment. This concern manufactures filter presses

| Type Engine. | Fuel. | Price. | Fuel Consumption per hp. Hour. | Cost per hp. Hour. | Cost per hp. Month. |
|---|---|---|---|---|---|
| Gasoline engine | Gasoline | 10 cents per gallon | ⅛ gallon | .0125 | 2.34 |
| Gas engine | Illuminating gas | $1 per 1,000 cubic feet | 18 cubic feet | .018 | 4.21 |
| Gas engine | Natural gas | 25 cents per 1,000 cu. feet | 13 to 15 cubic feet | .00325 to .00375 | .76 to .88 |
| Ordinary steam engine | Bituminous coal | $3 per ton | 6 to 10 lbs. | .009 to .015 | *2.10 to 3.51 |
| Producer gas engine | Anthracite pea coal | $4 per ton | 1 to 1¼ lbs. | .002 to .0025 | .47 to .59 |
| Producer gas engine | Gas coke | $5 per ton | 1¼ to 1½ lbs. | .0031 to .00375 | .73 to .80 |

*Cost of engineer and other attendance should be added to this; also large amount of water necessary.

Total amount of water used by producer is approximately 4 gallons per horse-power per hour.

**Jubilee of the Coal-Tar Industry.**—By a mistake of the printer the following passages were omitted from our report of the Jubilee of the Coal-Tar Industry. They should be inserted on the top of page 432: Dr. W. F. Hillebrand, president

for all classes of industrial filtration and also small presses for laboratory work.

**Public Lectures.**—Among the long list of free public lectures to be held in the season of 1906-7 under the auspices of

the Department of Education in the City of New York, we find the following: Prof. J. N. Gray, eleven lectures on General Physics; Prof. E. R. von Nardroff, eight lectures on Heat as a Mode of Motion; Dr. Charles Baskerville, on Radium and its Application; Mr. W. W. Ker, eleven lectures on Principles and Practice of Electrical Engineering; Prof. C. L. Harrington, eight lectures on Electricity; Prof. Morris Loeb, five lectures on the Chemistry of Carbohydrates; Prof N. A. DuBois, three lectures on Chemistry.

**Aluminium.**—According to the United States Geological Survey, the production of aluminium in this country was 61,281 pounds in 1890, 920,000 pounds in 1895, 7,150,000 pounds in 1900, and 11,347,000 in 1905. The production of bauxite in this country amounted to 48,129 gross tons in 1905, which is the maximum annual output ever achieved. In addition to the home production 11,726 gross tons of bauxite were imported during 1905.

**Electric Furnace for Hardening and Annealing Steel.**—In the London *Electrical Review*, Nov. 23, an electric furnace is described which was exhibited by S. E. Fedd at an electrical exhibition in Sheffield. It is specially adapted for hardening and annealing steel. It consists of a rectangular crucible built up of firebrick and clay within a cast iron casing. At opposite sides electrodes of wrought iron are fixed with massive connections well screened within the firebrick lining, which is of such thickness that the outer casing remains cool and but little loss of heat takes place. The bath consists of crystals of pure barium chloride, which is molten by the passage of the current, and can be made uniform at any temperatures between 750° and 1,400° C. For the higher temperatures above 1,100° C., barium chloride alone is used, but for the lower range it is necessary to mix it with potassium chloride. The cost of the charge is insignificant. Alternating currents are used with pressure regulation by means of a transformer. Experiments have shown that the temperature of the molten mass is perfectly uniform at every part of the bath and at any depth, with the exception of a layer from 10 to 15 millimeters thick at the top surface, which is in contact with the air. The rate of energy consumption is stated to vary from 4.10 watts at 750° C. to 49.2 watts at 1,300° C. per cubic inch of charge in furnaces of medium size. Over twenty of these furnaces are at present in operation, and it is stated that with gas at 30 cents per thousand cubic feet, and electric power at 2 cents per kilowatt-hour, the actual cost of hardening is about the same for the latest forms of gas ovens and for the electric furnace, and the output per month is greatly increased with the latter.

**Screening and Sizing.**—Bulletin 21 of the Colorado Iron Works Company describes in detail this company's patent "impact screen." The motion of this screen is perpendicular, or at right angles to the plane of the screen surface, while in all other flat screens the motion is nearly in the same plane in which the material is traveling. The motion of the impact screen, combined with the impact or vibration given to the screen cloth at each stroke, prevents small particles of ore from lodging in the meshes. By thus keeping the screen cloth open, its capacity is enormously greater than any form of revolving screen. Another advantage is the small space occupied. The impact screen of the Colorado Iron Works fills the requirements between the limits of about ¼ inch and 8 or 100-mesh, sizing pulp between these extremes in perfect manner. This screen was first brought out a few years ago, and has been improved in the meantime in some details. The same bulletin contains notes on cylindrical and conical revolving screens or trommels, on octagonal revolving screens, on hydraulic classifiers, on spitzkasten, and on the Dow classifier, which separates the sands from the slimes in cyanide mills.

**Niagara Falls.**—One of the strongest and ablest arguments in favor of industrial Niagara Falls is made in a letter of Mr. F. W. Haskell, president of the Carborundum Co., to Capt. Chas. W. Kutz, and recently published in the *Niagara Falls News*. The letter points out the misconceptions and misrepresentations which are at the bottom of the strong public sentiment against the industrial development of Niagara. He agrees with Mr. Buck that Niagara, as a mighty saver of labor and coal, is a vastly greater asset to the people than as a mere show place, "but while I have this belief I realize that it is Utopian." Under existing conditions the line will be drawn somewhere in the development of the waterpower. But Mr. Haskell insists that it should not and cannot honestly be drawn in such a way as to interfere with the contracted or expected power supply of any concern which has invested funds and developed an industry with the expectation, based on the good faith of the State of New York, that that investment would be protected and not strangled just as it attains maturity.

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## Digest of U. S. Patents.

*Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington D. C.*

OZONE AND MISCELLANEOUS GAS REACTIONS (*Concluded.*)

No. 652,081, June 19, 1900, J. W. Chisholm, of San Francisco, Cal.

Produces gas for heating and illumination by supplying air and steam to ignited carbonaceous fuel and conducting the products through a series of heated regenerator chambers, each filled with superposed crucibles containing "iron and copper or other material of opposite polarities." Said to be an improvement on Hall patents 494,198-9-200.

No. 659,236, Oct. 9, 1900, A. C. Johnson, of Baltimore, Md.

Passes electric sparks through the lead chambers of sulfuric acid plants, to precipitate the acid-laden moisture in suspension. The spark is said to have an effect in converting the steam and sulfur dioxid into sulfuric acid. Oxyhydrogen gas may be simultaneously introduced, the explosion assisting the precipitation.

No. 665,266, Jan. 1, 1901, F. Purdy, of Chicago, Ill.

Produces gas for fuel and illumination by passing air through ignited bituminous coal, mixing the product with oil and superheated steam, and passing through a chamber containing a filling of refractory material. An electrode having a series of horizontal iron shelves, embedded, except at the ends, in refractory material, depends centrally into the chamber, and a similar annular electrode of opposite polarity constitutes the lining of the chamber. The highly heated products of combustion passing through the refractory material are said to constitute a part of the electric circuit.

No. 667,099, Jan. 29, 1901, E. C. Paramore, of Philadelphia, Pa.

Treats chlorin gas to eliminate its odor, change its color and enhance its bleaching action, by passing the moist chlorin from a generator containing manganese dioxid and dilute hydrochloric acid, through a tube wherein it is subjected to an electric discharge. The chlorin is caused to move backward and forward through the tube by a pump, so as to be repeatedly subjected to the action of the electric current.

No. 667,100, Jan. 29, 1901, E. C. Paramore, of Philadelphia, Pa.

Apparatus for carrying out the process of the preceding patent. The glass tube through which the chlorin moves has lateral branches into which extend wire-brush electrodes, hermetically sealed in glass bulbs.

No. 671,507, April 9, 1901, R. J. Yarnold, of Streatham, England.

A number of spaced dielectric sheets, for example, of glass or mica, are interposed between two electrodes, the air or gas to be treated passing between them. The dielectrics and electrodes may be flat and arranged in a pile or tubular and concentric.